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Stibnite Gold Project

Prefeasibility Study Technical Report

Valley County, Idaho

Lee A. Becker, P.E.
Richard K. Zimmerman, R.G.
Garth D. Kirkham, P. Geo.
Christopher J. Martin, C.Eng.
John M. Marek, P.E.
Allen R. Anderson, P.E.
Richard C. Kinder, P.E.
Peter E. Kowalewski, P.E.

Prepared For:

MIDAS GOLD

DATE AND SIGNATURES PAGE

The effective date of this Report is December 8, 2014. The issue date of this Report is March 28, 2019. See Appendix I, Prefeasibility Study Contributors and Professional Qualifications, for certificates of Qualified Persons.

(Signed) *"Lee A. Becker"* Lee A. Bekcer, P.E.	March 28, 2019 Date
(Signed) *"Richard K. Zimmerman"* Richard K. Zimmerman, R.G.	March 28, 2019 Date
(Signed) *"Garth D. Kirkham"* Garth D. Kirkham, P.Geo.	March 28, 2019 Date
(Signed) *"Christopher J. Martin"* Christopher J. Martin, C.Eng.	March 28, 2019 Date
(Signed) *"John M. Marek"* John M. Marek, P.E.	March 28, 2019 Date
(Signed) *"Allen R. Anderson"* Allen R. Anderson, P.E.	March 28, 2019 Date
(Signed) *"Richard C. Kinder"* Richard C. Kinder, P.E.	March 28, 2019 Date
(Signed) *"Peter E. Kowalewski"* Peter E. Kowalewski, P.E.	March 28, 2019 Date

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1 SUMMARY

1.1 INTRODUCTION AND PURPOSE

Redevelopment of the Stibnite Gold Project (**Project**) has the potential to clean up an existing brownfield site, one that has been extensively mined for more than 80 years, and which could become one of the largest gold producers in the United States. This billion-dollar Project could create more than 700 jobs in Idaho during the first three years of construction and nearly 1,000 jobs in Idaho during 12 years of Project operations, while generating significant taxes and other benefits to the local, state and federal economies. This preliminary feasibility study (**PFS**) and related Technical Report (**Report**) provide a comprehensive overview of the Project and includes recommendations for future work programs required to advance the Project to a decision point. This Report defines an economically feasible, technically and environmentally sound Project that minimizes impacts and maximizes benefits.

Key considerations for the Project are as follows:

- The Project design began with the end in mind, contemplating the development, operation and closure of the Project on a sustainable basis, meeting society's present day needs for economic prosperity while remaining protective of the environment, and enhancing the ability of future generations to sustain their own needs.
- The Project is designed to ensure ongoing positive local and regional financial and social benefits through taxation, employment, and business opportunities in a region where the economy has suffered for more than a decade, resulting in some of the highest unemployment and lowest annual wages in Idaho.
- From the beginning, the Project has been designed for what will remain after closure. The plan for closure is protective of the environment and incorporates inherently stable, secure features that will provide the foundation for a naturally sustainable ecosystem.
- The Project design incorporates cleanup and repair of extensive historical mining-related impacts; much of the cleanup and repair would occur during initial construction and early operations.
- The new facilities contemplated for the Project are tightly constrained and, to a large extent, placed in historically impacted areas in order to minimize the incremental Project footprint.
- Salmon and other fishery enhancements are integral to the Project design. Removal of man-made barriers and reconstruction of natural habitat would allow salmon and other fish migration into the upper reaches of the watershed for the first time since 1938.
- During development, operations and closure, all aspects of the Project are designed to improve existing conditions, where possible, and remain protective of the environment, with the extensive costs related to remediation and reclamation of historical impacts accommodated by an economically feasible Project.

This Report provides information about the geology, mineralization, exploration potential, Mineral Resources, Mineral Reserves, mining method, process method, infrastructure, social and economic benefits, environmental protection, cleanup and repair of historical impacts, reclamation and closure concepts, capital and operating costs and an economic analysis for the Project.

1.2 KEY RESULTS

The Project consists of rehabilitating an existing brownfields site in an area of significant historical mining, including removal and reprocessing of the historic gold-silver-antimony tailings (**Historic Tailings**), and mining the Yellow Pine, Hangar Flats and West End gold-silver-antimony deposits using conventional open pit methods, conventional processing methods to extract gold, silver and antimony, and on-site production of gold (**Au**) and silver (**Ag**) doré and an antimony (**Sb**) concentrate. Midas Gold's plans for decommissioning the site include progressive and concurrent remediation and reclamation activities, beginning at the start of construction and continuing beyond the operations phase, through Project reclamation and closure.

The Stibnite Gold Project, as contemplated in the PFS, comprises:

- A design that minimizes Project footprint, and locates facilities within already impacted areas, and which incorporates a number of approaches to risk reduction, such as an improved access road to the site that avoids all major waterways.
- An extensive reclamation and remediation program for historical impacts to the site including, but not limited to, the recovery and reprocessing of Historic Tailings, restoration of fish passage during and after operations, removal of historical waste rock to an engineered waste rock storage facility, repair of the Blowout Creek channel that is a source of significant sedimentation, reforestation of impacted areas, stream channel repairs, etc. Many of these activities will occur during construction and/or relatively early in the mine life.
- Four deposits, including Historic Tailings, with combined Indicated Mineral Resources of 115.2 million short tons (**Mst**) or 104.5 million metric tonnes (**Mt**) grading 0.048 troy ounces per short ton (**oz/st**) Au or 1.63 grams per metric tonne (**g/t**) Au, 0.077 oz/st (2.65 g/t) Ag, and 0.07% Sb. The aggregate Indicated Mineral Resources contain 5.46 million oz (**Moz**) Au, 8.90 Moz Ag, and 155.2 million pounds (**Mlbs**) Sb.
- Combined Probable Mineral Reserves of 98.07 Mst (88.96 Mt) grading 0.047 oz/st (1.60 g/t) Au, 0.071 oz/st (2.43 g/t) Ag, and 0.07% Sb. Total contained metal in the Probable Mineral Reserves includes 4.58 Moz Au, 6.96 Moz Ag, and 137.0 Mlbs Sb.
- The four deposits combined also contain additional Inferred Mineral Resources of 10.8 Mst (9.8 Mt) grading 0.032 oz/st (1.10 g/t) Au, 0.049 oz/st (1.67 g/t) Ag, and 0.04% Sb that is not utilized in the PFS. The combined Inferred Mineral Resources contain 347 koz Au, 523 koz Ag, and 9.5 Mlbs Sb. *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. These Mineral Resource estimates include Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated.*
- The mineralization is primarily hosted in sulfides, with modest amounts of oxides, both of which can be treated with different extraction processes in the same plant. Sulfide mineralization would be milled and treated with bulk flotation, or with sequential flotation when sufficient antimony is present, to produce two products: (1) an antimony concentrate for off-site shipment to a smelter, and (2) a gold concentrate. In both cases, sulfide gold concentrates would be further processed on-site using pressure oxidation (**POX**) followed by vat leaching to produce gold-silver doré. The oxide material would be milled and then vat leached to recover gold and silver only, with a significant portion processed during down times for the POX circuit.
- Production is recommended to average 8.05 Mst of ore fed to the crusher per year (22,050 short tons per day (**st/d**)) with an average strip ratio of 3.5:1; in metric units, this equates to 7.30 Mt and 20,000 t/d. With this production rate, the mine life would be approximately 12 years. The average mill feed gold grade for the Project is approximately 0.047 oz/st (1.60 g/t) containing 4,575 thousand ounces (**koz**) of gold with significant sliver and antimony credits.
- Payable metals for the Project total 4,006 koz of gold, 1,467 koz of silver, and 67,900 thousand lbs (**klbs**) of antimony for the Project life of mine.
- Total capital cost would be approximately $1,125 million, including start-up capital costs of $970 million, sustaining capital costs of $99 million, and closure costs of $56 million.

- Using the Base Case economic factors detailed in Section 22, the financial model yields a pre-tax net present value at a 5% discount rate ($PTNPV_{5\%}$) of $1,093 million and an after tax net present value at a 5% discount rate ($ATNPV_{5\%}$) of $832 million. As currently designed, the Project's Internal Rate of Return (IRR) is 19.3% with a payback period of approximately 3.4 production years.
- The $ATNPV_{5\%}$ for the Project is most sensitive to changes in revenue, which is manifested as changes in metal prices, gold grades, or gold recovery. For example, a 20% increase in gold price or gold grade raises the $ATNPV_{5\%}$ from $832 million to $1,369 million, a 63% increase for the base Case. Similarly, a decrease of 20% in gold grade, gold recovery, or gold price results in a 71% decrease in $ATNPV_{5\%}$ for the Base Case.
- A number of risks and opportunities have been identified, including the potential for additional gold production from within the current pit outlines, as well as some from outside, either of which could significantly enhance the economic outcomes for the Project.
- The closure concept for the Project envisions removal or demolishing of onsite facilities, comprehensive reclamation and reforestation of disturbed areas, permanent establishment of fish passage through the site, and a sustainable environment.

The economic and technical analyses included in this Report provide only a summary of the potential Project economics based on the assumptions set out herein. There is no guarantee that the Project economics described herein can be achieved.

1.3 REGULATORY INFORMATION

This Report has been prepared based on the results of a PFS completed for the Project, which is located in the Stibnite-Yellow Pine mining district (**District**), Idaho. The Project is wholly owned by direct or indirect subsidiaries of Midas Gold Corp. ("**MGC**"), a TSX-listed British Columbia company. Unless the context indicates otherwise, references to "**Midas Gold**" throughout this Report include one or more of the aforementioned subsidiaries of MGC.

This Report has been prepared under the direction of Independent Qualified Persons (**QP**) and in compliance with the Canadian Securities Administrators (**CSA**) National Instrument 43-101 (**NI 43-101**) standards for reporting mineral properties, Companion Policy 43-101CP, and Form 43-101F1. This Report supersedes and replaces the technical report entitled '*Preliminary Economic Assessment Technical Report for the Golden Meadows Project, Idaho'* prepared by SRK Consulting (Canada) Inc. and dated September 21, 2012 (**PEA**) and that report should no longer be relied upon.

For readers to fully understand the information in this Report, they should read the Report (to be available on SEDAR or at www.midasgoldcorp.com by the end of 2014) in its entirety, including all qualifications, assumptions and exclusions that relate to the information set out in this Report that qualifies the technical information contained in the Report. The Report intended to be read as a whole, and sections should not be read or relied upon out of context. The technical information in the Report is subject to the assumptions and qualifications contained in the Report.

1.4 RELIANCE ON OTHER EXPERTS

Certain sections of the Report rely on reports and statements from legal and technical experts who are not Qualified Persons as defined by NI 43-101. The Qualified Persons responsible for preparation of this Report have reviewed the information and conclusions provided that they are responsible for and determined that they conform to industry standards, are professionally sound, and are acceptable for use in this Report.

1.5 PROPERTY DESCRIPTION AND LOCATION

The Stibnite Gold Project is located in central Idaho, USA. The Project lies approximately 100 miles (mi) northeast of Boise, Idaho, 38 mi east of McCall, Idaho, and approximately 10 mi east of Yellow Pine, Idaho. Figure 1.1 illustrates the location of the Project.

The Hangar Flats, West End, and Yellow Pine deposits, along with the Historic Tailings, lie within mineral concessions controlled by Midas Gold, as are other exploration prospects and targets identified in this Report. Mineral rights controlled by Midas Gold include patented lode claims, patented mill site claims, unpatented federal lode claims, and unpatented federal mill site claims and encompass approximately 27,104 acres or 42 square miles. The claims are 100% owned, except for 27 patented lode claims that are held under an option to purchase. The Project is subject to a 1.7% NSR Royalty on gold only. There is no royalty on silver or antimony.

1.6 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Stibnite Gold Project is located approximately 152 road-miles northeast of Boise, Idaho in an area of deeply incised drainage related to the East Fork of the South Fork of the Salmon River (EFSFSR) at an elevation of ~ 6,500 feet (ft) with nearby mountains rising to an elevation of approximately 7,800 to 8,900 ft.

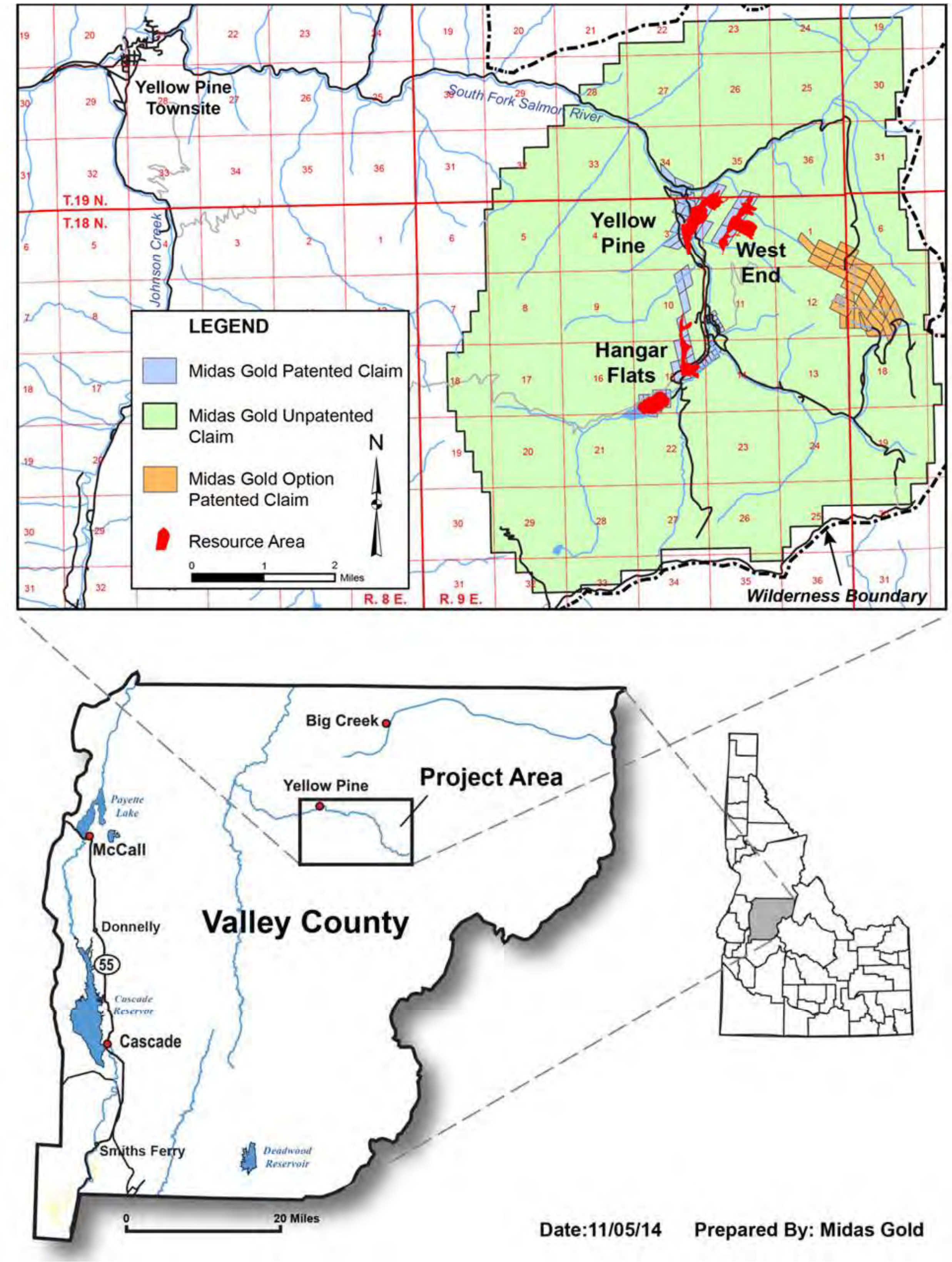
The climate is characterized by moderately cold winters and mild summers. Most precipitation occurs as snowfall in the winter and rain during the spring. The local climate allows for year round operations, as evidenced by historical production over extended periods, and climate information.

Ground access to the Property is currently available by road from the nearby towns of Cascade, Idaho, an 84 mile drive and, during the snow free months, from McCall, Idaho, which is a 63-mi drive. The closest rail is in Cascade, while the closest access for sea transportation is on the west coast of the US and Canada, or via the inland port of Lewiston, ID.

Power-lines would need to be installed/upgraded from the main regional Idaho Power Corporation (IPCo) substation at Lake Fork to the Project site, a distance of 42 mi, along an existing and previously used right-of-way.

Midas Gold has four permanent and three temporary water rights in the District.

Figure 1.1: Location Map of the Stibnite Gold Project



1.7 HISTORY

The Project is located in a past-producing area near the historical town of Stibnite. Since the late 1920s, gold, silver, antimony, tungsten, and mercury mineralized materials have been mined in the area by both underground and, later, open pit methods, creating numerous open pits, underground workings, large-scale waste rock dumps, heap leach pads, spent heap leach ore piles, tailings depositories, a mill site, three town sites, an airstrip, and other disturbances, some of which still exist today. Antimony-tungsten-gold sulphide milling operations ceased in 1952 as a result of lower metal prices following the end of the Korean War, while mercury operations on the Cinnabar claims continued until 1963. Exploration recommenced in 1974, followed by open pit mining and seasonal on-off heap leaching from 1982 to 1997. Midas Gold commenced its exploration activities in 2009.

Table 1.1 summarizes the approximate historical production for the Project by area; additional details are provided in Section 6.

Table 1.1: Estimated Historical Metal Production

Area	Production Years	Tons Mined (st)	Recovered Au (oz)	Recovered Ag (oz)	Recovered Sb (st)	Recovered WO_3 (units)(1)
Hangar Flats	1928 - 38	303,853	51,610	181,863	3,758	67
Yellow Pine	1938 - 92	6,493,838	479,517	1,756,928	40,257	856,189
West End	1978 - 97	8,156,942	454,475	149,760	-	-
Totals		14,954,633	985,602	2,088,551	44,015	856,256

Note:
(1) A unit of WO_3 (tungsten trioxide) is 1% of a short ton (20 pounds), and WO_3 is 79.3% tungsten. A short ton unit of WO_3, therefore, equals 20 pounds of WO_3 and contains 15.86 pounds of tungsten.

1.8 GEOLOGICAL SETTING AND MINERALIZATION

The Project area is underlain by pre-Cretaceous “basement” sediments, the Cretaceous-age Idaho Batholith (granitic), Tertiary-age intermediate to felsic intrusions and volcanics, younger unconsolidated sediments derived from erosion of the older sequences and glacial materials.

Large, north-south striking, steeply dipping to vertical structures exhibiting pronounced gouge and multiple stages of brecciation occur in the central and eastern portions of the property and are often associated with east-west and northeast-southwest trending splays and dilatant structures.

Intrusive-hosted precious metals mineralization typically occurs in structurally prepared zones in association with very fine-grained disseminated arsenical pyrite (FeS_2) and, to a lesser extent, arsenopyrite (FeAsS), with gold almost exclusively in solid solution in these minerals.

Antimony mineralization occurs primarily associated with the mineral stibnite (Sb_2S_3). Zones of silver-rich mineralization locally occur with antimony and are related to the presence of pyrargyrite (Ag_3SbS_3), hessite (Ag_2Te) and acanthite (Ag_2S).

Metasediment-hosted mineralization has a similar sulfide suite and similar geochemistry to the intrusive hosted mineralization, but with higher carbonate content in the gangue and a much more diverse suite of late stage minerals.

1.9 DEPOSIT TYPES

The origin of the wide variety of mineralization occurrences at the Stibnite Gold Project is attributed to deep-seated intrusives and associated high temperature and high pressure processes to shallow lower temperature, lower pressure hydrothermal processes.

1.10 EXPLORATION

The District has been the subject of exploration and development activities for nearly 100 years. Numerous prospects have been discovered through the years using a variety of methods. Some of these prospects were developed into mines and others remain undeveloped; further, new ones may be discovered as the Project advances and the nature of mineralization previously exploited is better understood.

Midas Gold's analysis of historical data and its exploration since 2009 has identified a number of key exploration opportunities:

- There is potential at each of the Hangar Flats, West End and Yellow Pine deposits to increase Mineral Resources and Mineral Reserves at grades higher than cut-off, this potential includes conversion of currently Inferred Mineral Resources to higher confidence levels, conversion of currently unclassified material within the economic pits, and expansion potential immediately adjacent to the existing Mineral Resources and Mineral Reserves that could result in increased Mineral Reserves and reduced strip ratios;
- There is good potential to delineate high grade, Au +/- Sb, near surface underground mineral deposits at prospects such as Scout, Garnet and Upper Midnight (based on varying degrees of drilling already completed) that could provide supplemental early mine life, higher margin, mill feed;
- There is potential for the discovery and definition of additional mineral deposits along the main mineralized trends, such as between Hangar Flats and Yellow Pine, based on exploration and drilling completed to date;
- A number of other prospects have been defined to varying degrees, up to and including detailed drilling, that indicate potential for bulk tonnage disseminated Au deposits similar to those containing the current Mineral Resources – these include the Rabbit and Ridgetop-Cinnamid prospects; and
- A number of prospects, such as Mule, have different geologic settings to those discussed above but which could potentially develop into significant mineral deposits.

Note: There has been insufficient exploration to define Mineral Resources on these prospects and it is uncertain whether further exploration will result in the targets being delineated as either Mineral Resources or Mineral Reserves.

1.11 DRILLING

The Project area, including the three main deposits, has been drilled by numerous operators, totaling 773,744 ft in 2,606 drill holes, of which Midas Gold drilled 550 holes, totaling over 326,275 ft, since 2009. Pre-Midas Gold drilling was undertaken by a wide variety of methods and operators while Midas Gold employed a variety of drilling methods including core, Reverse Circulation, auger, and sonic throughout the District, but with the primary method being core. All Midas Gold holes were surveyed and recoveries were generally good to excellent. Industry standard QA/QC procedures were used by Midas Gold, including sample security, blanks, standards and duplicates and these procedures were verified by the Independent QP.

1.12 DATA VERIFICATION

Extensive data verification programs have been undertaken by numerous independent consultants for Midas Gold and by Midas Gold personnel, as discussed in previous NI 43-101 technical reports (SRK, 2011; SRK, 2012) and discussed in this Report. These verification programs have been essential in ensuring that the datasets used for the Mineral Resource estimates are validated and verified as adequate for the estimation of Mineral Resources for each of the respective deposits. It is the opinion of the Independent QP responsible for the Mineral Resource estimates that the data used for estimating the Mineral Resources and Mineral Reserves for the Hanger Flats, West End, Yellow Pine and Historic Tailings deposits is adequate for this purpose and may be relied upon to report the Mineral Resources and Mineral Reserves contained in this Report.

1.13 MINERAL PROCESSING AND METALLURGICAL TESTING

Subsequent to the test work program undertaken for the 2012 PEA and other historical testing undertaken by prior owners and operators, a total of seven flowsheet development composites and 114 variability composites were prepared for metallurgical testing in support of the PFS from the more than 800 samples collected from the Project. Mineralogical work confirmed that the gold is mostly present in both pyrite and (to a much lesser extent) arsenopyrite, at concentrations that are usually high enough to economically justify flotation concentration followed by POX of the sulfides and cyanidation of the released gold. Oxide zones, mostly in the West End Deposit, contained very fine-grained, discrete gold available to direct cyanidation. Antimony occurs as stibnite, which is typically coarse-grained when occurring in higher-grade samples.

After the PEA related testing, grindability testing was conducted on all deposits, including two JK Drop Weight tests, 22 JK SAG mill characterization (SMC) tests, 10 crusher work index and abrasion index tests, 8 rod mill work index, and 24 ball mill work index tests. All composites indicate medium hardness (ball mill work index 13.0 to 14.1 kWh/t) and are amenable to semi-autogenous grinding (SAG) milling, though West End is somewhat more resistant to SAG milling, and Yellow Pine appears to be slightly more resistant to ball milling.

Over 300 metallurgical tests were completed on samples from the Yellow Pine, Hangar Flats, West End and Historic Tailings deposits as part of the PFS; in addition, more than 130 tests were completed for the PEA and numerous test programs were completed by prior owners and operators. Despite some mineralogical differences between the deposits, developmental metallurgical testwork has been able to identify a single, modular flowsheet that proved successful when applied to each of the deposits, making it possible to design a single plant that can process all ores from the Project as they are mined. This plant would, when antimony grades are high enough, float off the stibnite to create a saleable antimony concentrate, and then all ores (whether or not antimony is pre-floated) would be subject to bulk flotation of sulfides to produce an auriferous concentrate. Limited testwork on the Historic Tailings showed that they could be successfully co-processed through either flowsheet with the early production Yellow Pine ores.

At most times, the rougher flotation concentrates are expected to meet the POX sulfur content requirements and not require further cleaning, although West End concentrates require additional processing to reject carbonate-bearing (CO_3) minerals from the gold concentrates to produce a POX friendly concentrate.

Developmental leaching test work was also undertaken on the West End oxide ores, as well as on select flotation tailings produced from partially oxidized mineralization from Hangar Flats and West End. West End oxide leach studies indicate that 96% of the extracted gold leaches in the first six hours, with another 2% leached over the final 18 hours. Leach studies on the flotation tailings from Hangar Flats and West End indicate that any leachable gold in the flotation tailings is also fast leaching and could contribute significantly to gold recovery. Leach studies on the flotation tailings from Yellow Pine suggest little incremental recovery, but leaching them would provide additional assurance against losses of cyanide-soluble gold.

The projected overall recoveries for each deposit are shown on Figure 1.2 and Figure 1.3.

Figure 1.2: Gold and Silver Recoveries to Doré

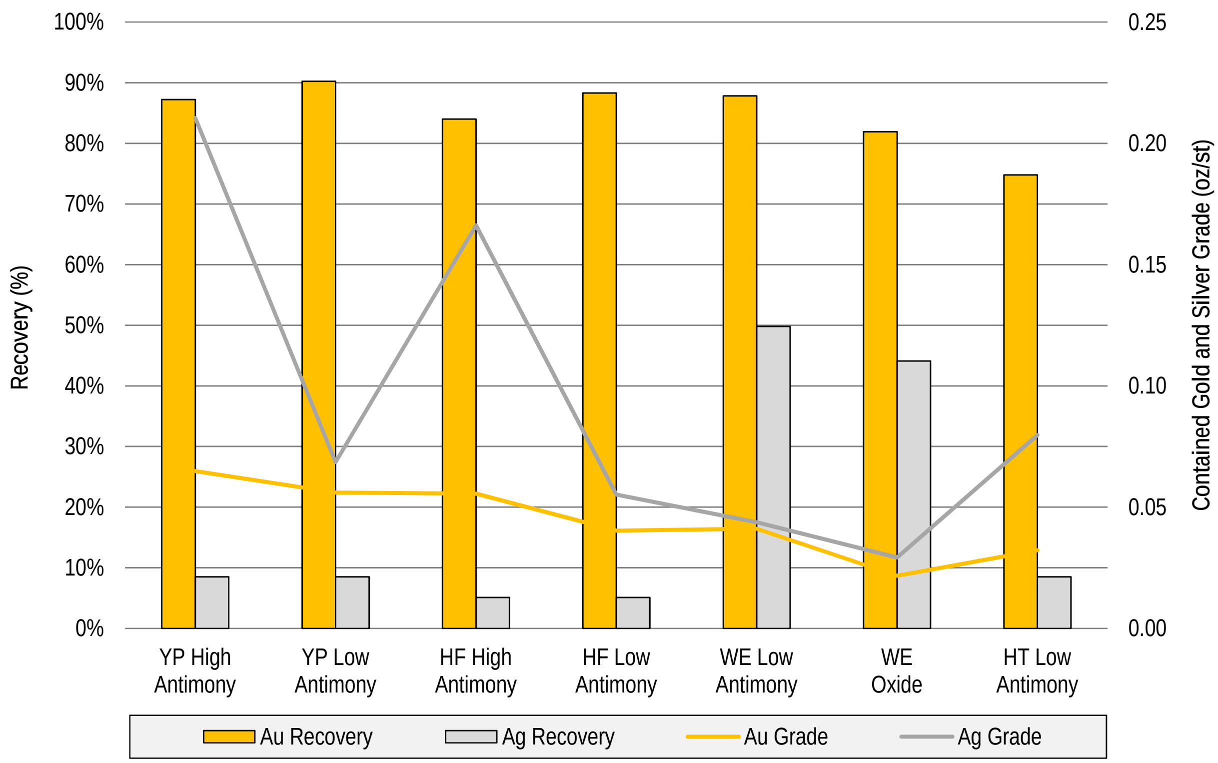
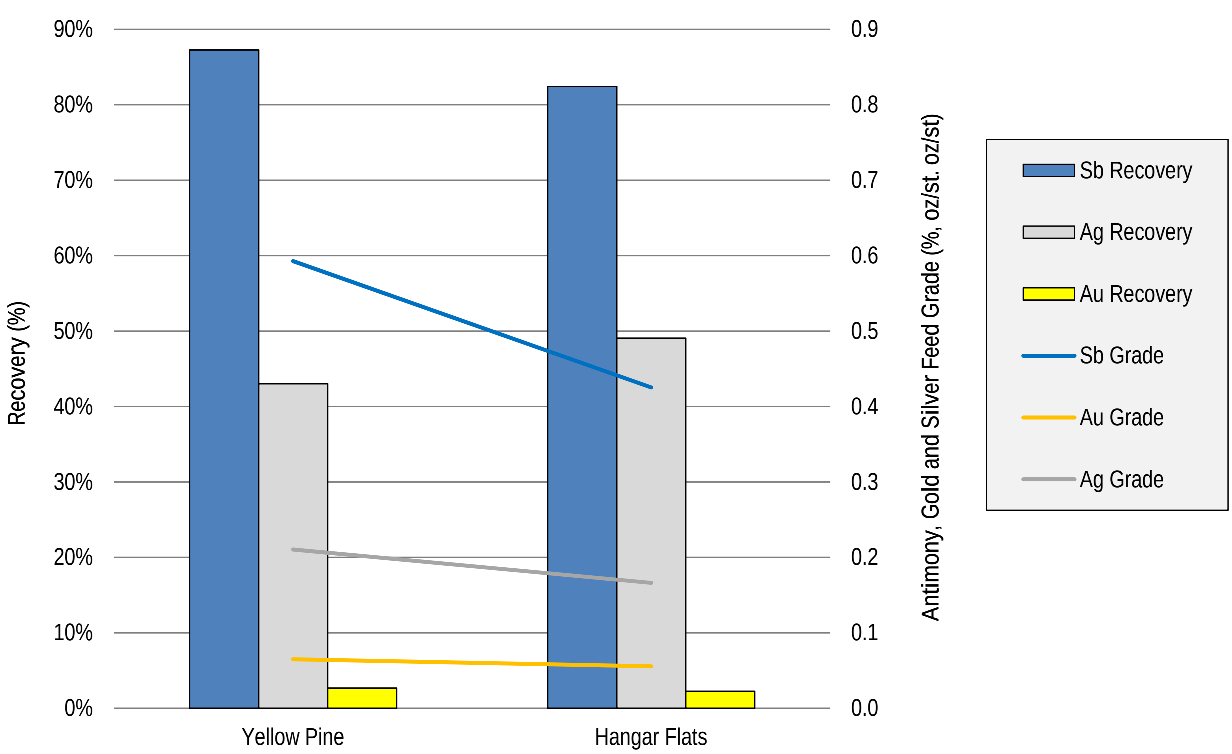


Figure 1.3: Antimony Concentrate Recoveries



1.14 MINERAL RESOURCE ESTIMATES

The Mineral Resource estimates for Hangar Flats, West End and Yellow Pine deposits, and the Historic Tailings, were prepared to industry standards and best practices using commercial mine-modeling and geostatistical software by third party consultants and verified by an Independent QP.

The Mineral Resources were initially calculated using a gold price of $1,400/oz and parameters defined in Section 14; based on this, the open pit sulfide cut-off grade was calculated as approximately 0.016 oz/st (0.55 g/t) Au and the open pit oxide cut-off grade calculated as approximately 0.010 oz/st (0.35 g/t) Au. However, Midas Gold elected to report its Mineral Resources at a 0.022 oz/st (0.75 g/t) Au sulfide cut-off grade and 0.013 oz/st (0.45 g/t) Au oxide cut-off grade, which is equivalent to utilizing the cost assumptions stated in Section 14 and a gold selling price of approximately $1,000/oz for sulfides and $1,100/oz for oxides. The consolidated Mineral Resource statement for the Project is shown in Table 1.2.

Table 1.2: Consolidated Mineral Resource Statement for the Stibnite Gold Project

Classification	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated							
Hangar Flats	21,389	1.60	1,103	4.30	2,960	0.11	54,180
West End	35,974	1.30	1,501	1.35	1,567	0.008	6,563
Yellow Pine	44,559	1.93	2,762	2.89	4,133	0.09	84,777
Historic Tailings	2,583	1.19	99	2.95	245	0.17	9,648
Total Indicated	*104,506*	*1.63*	*5,464*	*2.65*	*8,904*	*0.07*	*155,169*
Inferred							
Hangar Flats	7,451	1.52	363	4.61	1,105	0.11	18,727
West End	8,546	1.15	317	0.68	187	0.006	1,083
Yellow Pine	9,031	1.31	380	1.50	437	0.03	5,535
Historic Tailings	140	1.23	6	2.88	13	0.18	563
Total Inferred	*25,168*	*1.32*	*1,066*	*2.15*	*1,743*	*0.05*	*25,908*

Notes:

(1) All Mineral Resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum ("CIM") definitions, as required under National Instrument 43-101 ("NI43-101").

(2) Mineral Resources are reported in relation to a conceptual pit shell in order to demonstrate potential for economic viability, as required under NI43-101; mineralization lying outside of these pit shells is not reported as a Mineral Resource. ***Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. These Mineral Resource estimates include Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated.*** *All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely.*

(3) Open pit sulfide Mineral Resources are reported at a cutoff grade of 0.75 g/t Au and open pit oxide Mineral Resources are reported at a cutoff grade of 0.45 g/t Au.

The Yellow Pine and Hangar Flats deposits contain zones with substantially elevated antimony-silver mineralization, defined as containing greater than 0.1% antimony, relative to the overall Mineral Resource. The existing Historic Tailings Mineral Resource also contains elevated concentrations of antimony. These higher-grade antimony zones are reported separately in Table 1.3. Antimony zones are reported only if they lie within gold Mineral Resource estimates.

Table 1.3: Antimony Sub-Domains Consolidated Mineral Resource Statement

Classification	Tonnage (kt)	Gold Grade (g/t)(3)	Contained Gold (koz)	Silver Grade (g/t)(3)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Total Indicated	12,564	1.98	800	6.23	2,518	0.50	138,218
Total Inferred	1,735	1.74	97	6.88	384	0.60	22,959

Notes:
(1) *Antimony Mineral Resources are reported as a subset of the total Mineral Resource within the conceptual pit shells used to constrain the total Mineral Resource in order to demonstrate potential for economic viability, as required under NI43-101; mineralization outside of these pit shells is not reported as a Mineral Resource.* ***Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. These Mineral Resource estimates include inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated.*** *All figures are rounded to reflect the relative accuracy of the estimate.*
(2) *Open pit antimony sulfide Mineral Resources are reported at a cutoff grade 0.1% antimony within the overall 0.75 g/t Au cutoff.*
(3) *Includes contributions from Hangar Flats, Yellow Pine and Historic Tailings. See Section 14 for details.*

1.15 MINERAL RESERVE ESTIMATES

The qualified person (QP) for the estimation of the Mineral Reserve was John M. Marek, P.E. of Independent Mining Consultants, Inc. The Mineral Reserves were estimated in conformity with generally accepted Canadian Institute of Mining and Metallurgy (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" and are reported in accordance with the Canadian Securities Administrators' NI 43-101. Mr. Marek has reviewed the risks, opportunities, conclusions and recommendations summarized in Sections 25 and 26, and he is not aware of any unique conditions that would put the Stibnite Gold Mineral Reserve at a higher level of risk than any other North American developing projects.

The Mineral Reserve was developed by allowing only Indicated Mineral Resource blocks to contribute positive economic value, and is a subset of the Mineral Resource comprised of the Probable Mineral Reserve that is planned for processing over the life-of-mine plan, with assumptions summarized in Sections 15 and 16. No economic credit has been applied to Inferred mineralization in the development of the Mineral Reserve; further blocks needed to be economic based on gold content alone before being categorized as a Mineral Reserve. A series of floating cones were developed by varying the gold price from $200/oz to $1,500/oz and then evaluated at a $1,200/oz price for gold without changing the size of the cone; for Yellow Pine, an $800/oz cone was selected as optimal, while $1,100/oz cones were selected for Hangar Flats and West End.

Based on the longer-term nature of the Project, cutoff grades for Mineral Reserves were developed assuming long term metal prices of $1,350/oz gold, $22.50/oz silver, and $4.50/lb antimony for material lying within the cones selected above. Confidence classification was based on gold estimation.

The cut-off grade is defined by a term called "Net of Process Revenue" (NPR) which takes into account final PFS processing recoveries, processing costs, and smelter terms (see Section 15), with any block with a NPR greater than zero meets the requirement for internal cutoff grade. The processing costs for ore range from $9.07/st for oxides to $17.00/st for high antimony sulfides with an additional $3.40/st of ore for G&A. Therefore the NSR equivalent of the cut-off grade range is: $12.47/st – $20.40/st. The Mineral Reserves are summarized in Table 1.4.

Table 1.4: Stibnite Gold Project Probable Mineral Reserve Estimate (Imperial & Metric Units)

Deposit	Tonnage	Average Grade			Total Contained Metal		
		Gold	Antimony	Silver	Gold	Antimony	Silver
Imperial Units	(kst)	(oz/st)	(%)	(oz/st)	(koz)	(klbs)	(koz)
Yellow Pine	43,985	0.057	0.098	0.090	2,521	86,376	3,973
Hangar Flats	15,430	0.045	0.132	0.086	690	40,757	1,327
West End	35,650	0.035	0.000	0.040	1,265	-	1,410
Historic Tailings	3,001	0.034	0.165	0.084	102	9,903	252
Total Probable Mineral Reserve(1)	98,066	0.047	0.070	0.071	4,579	137,037	6,962
Metric Units	(kt)	(g/t)	(%)	(g/t)	(t)	(t)	(t)
Yellow Pine	39,903	1.97	0.098	3.10	78.4	39,179	123.6
Hangar Flats	13,998	1.53	0.132	2.95	21.5	18,487	41.3
West End	32,341	1.22	0.000	1.36	39.3	-	43.9
Historic Tailings	2,722	1.17	0.165	2.88	3.2	4,492	7.8
Total Probable Mineral Reserve(1)	88,964	1.60	0.070	2.43	142.4	62,159	216.5

Notes:
(1) *Metal prices used for Mineral Reserves: $1350/oz Au, $22.50/oz Ag, $4.50/lb Sb.*
(2) *Block MUST be economical based on gold value only in order to be included as ore in Mineral Reserve.*
(3) *Numbers may not add exactly due to rounding.*

Mineral Reserves exclude approximately 10.8 Mst with average grades of 0.032 oz/st (1.10 g/t) Au, 0.049 oz/st (1.67 g/t) Ag and 0.05% Sb that are Inferred Mineral Resources that lie within the Mineral Reserve pit limits; conversion of some or all of these tons would increase payable metal and reduce strip ratios. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated.

1.16 MINING

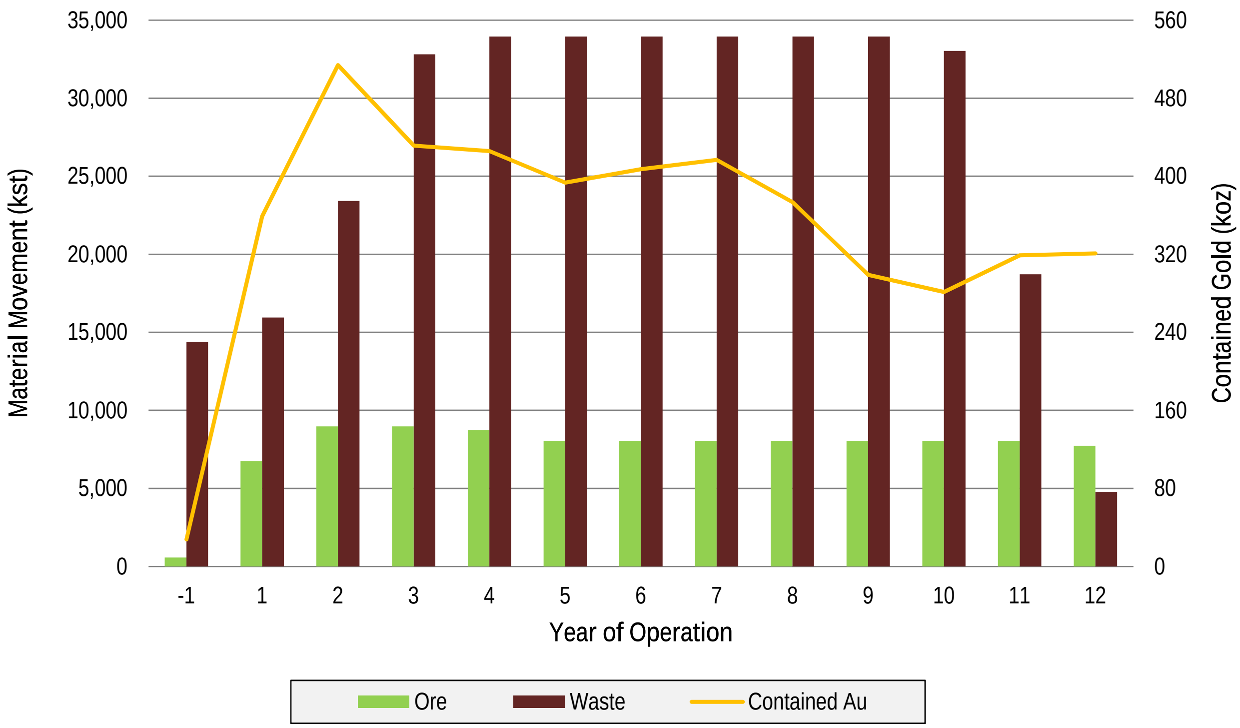
The mine plan developed for the Project incorporates the mining of the three *in situ* Mineral Deposits: Yellow Pine, Hangar Flats, and West End and their related waste rock, and the re-mining of Historic Tailings along with its cap of spent heap leach ore (SODA). Ore from the three pits would be sent to a centrally located crusher while the Historic Tailings would be fed by slurry into the process plant's grinding circuit. Waste rock would be sent to four distinct destinations: the tailings storage facility (TSF), the main Waste Rock Storage Facility (Main WRSF), the West End Waste Rock Storage Facility (West End WRSF), and to the Yellow Pine pit as backfill. The general sequence of mining would be the Yellow Pine deposit first, Hangar Flats second, and West End third. This planned sequence is driven by the need to backfill the Yellow Pine pit with waste rock from the West End pit in order to restore the original gradient of the EFSFSR while using environmentally appropriate carbonate-rich material for such backfill. This order generally follows a sequence of mining gold ounces from highest grade to lowest grade, and lowest cost to highest cost. The Historic Tailings, which lie within the footprint of the Main WRSF, would be removed during the first four years of the mine schedule to make the necessary space for the Main WRSF.

Mining at the Stibnite Gold Project would be accomplished using conventional open pit hard rock mining methods. Mining is planned to deliver 8.05 Mst of ore to the crusher per year (22,050 st/d), with stockpiling by ore type (low antimony sulfide, high antimony sulfide and oxide). Batches of oxide and sulfide material would be sent to the crusher; the oxide feed would be vat leached while the sulfide material would be floated to produce up to two concentrates: (1) an antimony concentrate, when there is sufficient antimony to justify recovering it, to be sent offsite and (2) a gold-bearing sulfide concentrate that would be oxidized in an autoclave and then sent to agitated leach tanks for gold-silver leaching.

The PFS mine plan schedules 98.066 Mst of ore to be fed to the processing plant from Yellow Pine, Hangar Flats and West End pits. The mining sequence requires the waste stripping to average 3.5:1 (waste rock: ore) for the first 3 years; then the stripping ratio would grow to 4.2:1 for years 4 through 9 after which it would drop to an average of 2.4:1 for the final 3 years. During the first four years, 3.0 Mst of Historic Tailings would be fed to the processing plant at a stripping ratio of 2.0:1 (SODA:tailings). The life-of-mine (LOM) strip ratio averages 3.5:1.

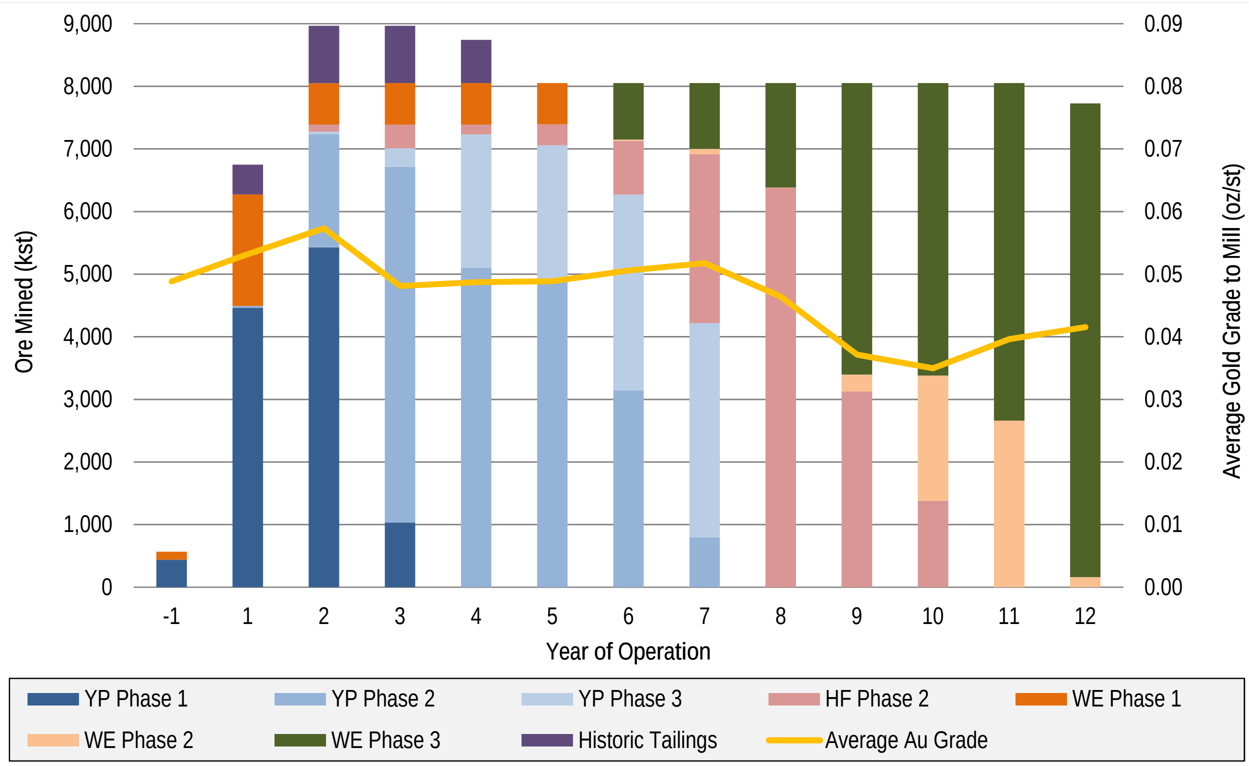
Figure 1.4 is a graphical depiction of the ore and waste rock movements from the mining phases by period and the contained gold ounces for the potential mine schedule for the Stibnite Gold Project; preproduction material from Year -1 would be processed in Year 1.

Figure 1.4: District Ore and Waste Movements and Ounces of Contained Gold Mined by Year



A summary of the mill feed by deposit is provided on Figure 1.5. This figure represents the Mineral Reserve because the Probable Mineral Reserve corresponds to the total ore processed in the mine.

Figure 1.5: Ore Mining Schedule by Deposit and Phase



A summary of the mill feed statistics by ore type is provided in Table 1.5

Table 1.5: LOM Mill Feed Statistics by Ore Type

Item	Unit	Value
General LOM Production Statistics		
Waste Rock Mined	Mst	346.7
Ore Mined	Mst	98.1
Strip Ratio (waste rock tons : ore tons).	st:st	3.5:1
Daily Mill Throughput	st/d	22,050
Annual Mill Throughput	Mst	8.05
Mine Life	production years	12
LOM Average Mill Head Grade		
Tonnage Milled	Mst	98.1
Gold Feed Grade	oz/st Au	0.047
Silver Feed Grade	oz/st Ag	0.071
Antimony Feed Grade	% Sb	0.070
Oxide Ore		
Tonnage Milled	Mst	10.7
Gold Feed Grade	oz/st Au	0.025
Silver Feed Grade	oz/st Ag	0.030
Antimony Feed Grade	% Sb	-

Item	Unit	Value
High Antimony Ore		
Tonnage Milled	Mst	11.0
Gold Feed Grade	oz/st Au	0.061
Silver Feed Grade	oz/st Ag	0.193
Antimony Feed Grade	% Sb	0.528
Low Antimony Ore (includes Historic Tailings)		
Tonnage Milled	Mt	76.3
Gold Feed Grade	oz/st Au	0.048
Silver Feed Grade	oz/st Ag	0.059
Antimony Feed Grade	% Sb	0.014

Mining would be performed with up to eighteen 200 st class haul trucks loaded by up to four 23.5 cubic yard front end loaders. The trucks would be light-body versions with an actual haulage capacity of 220 st. Blast holes would be 7-7/8" in diameter drilled by up to four drill rigs. An auxiliary fleet comprising dozers, motor graders water trucks and other ancillary equipment is also included in equipment requirements.

The overall gold recoveries to doré are expected to average approximately 90% from Yellow Pine, 87% from Hangar Flats, 86% from West End, and 75% from the Historic Tailings. When processing material containing more than 0.1% Sb, antimony recoveries are expected to average 82% for Hangar Flats and 87% for Yellow Pine, with minor gold and silver contained in the antimony concentrate.

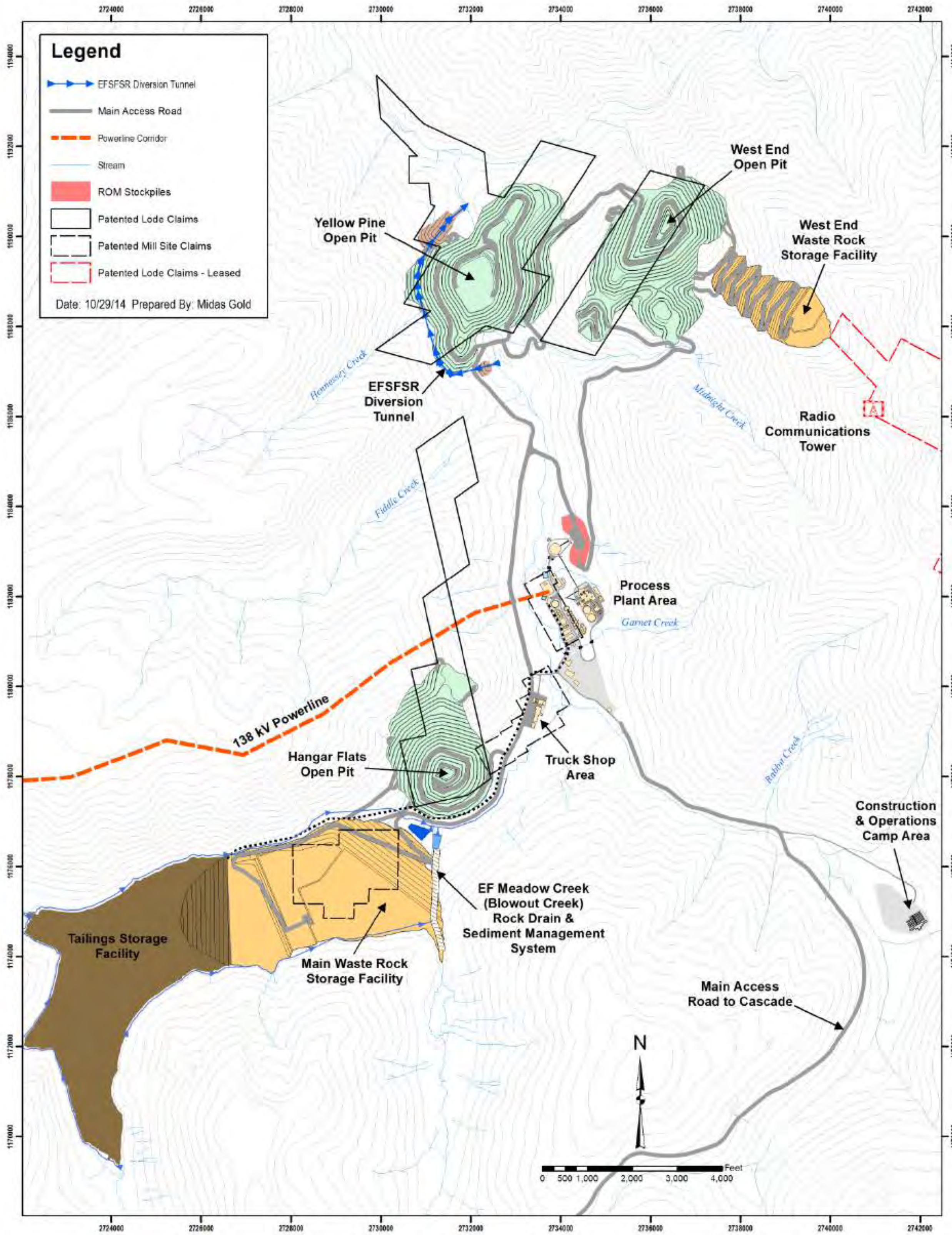
Figure 1.6 is a general overview of the mine site at the end of mine life prior to closure and reclamation.

1.17 RECOVERY

The Project's process plant has been designed to process sulfide, transition and oxide material from the Yellow Pine, Hangar Flats, and West End deposits. The processing facility is designed to treat an average of 22,050 st/d, or 8.05 Mst/y. Additionally, the Historic Tailings would be reprocessed early in the mine life to recover precious metals and antimony, and to provide space for the Main WRSF.

The overall gold recoveries to doré are expected to average approximately 90% from Yellow Pine, 87% from Hangar Flats, 86% from West End, and 75% from the Historic Tailings. When processing material containing more than 0.1% Sb, antimony recoveries are expected to average 82% for Hangar Flats and 87% for Yellow Pine, with minor gold and silver contained in the antimony concentrate.

Figure 1.6: Overall Site Layout



1.18 PROCESS OPERATION COMPONENTS

Run-of-mine (ROM) material would be crushed and milled, then flotation would be used to recover antimony as a stibnite flotation concentrate (with some silver and minor gold) when there is sufficient antimony to justify it. For all sulfide ore, an auriferous bulk sulfide flotation concentrate would be produced and oxidized in an autoclave. The autoclave residue and flotation tailings would be processed through conventional cyanidation and, doré bars produced containing gold and silver. Historic Tailings would be introduced into the ball mill during the first 3 - 4 years of operation. Tailings from the operation would be deposited in a geomembrane-lined TSF. The process operations include the following components:

- Crushing Circuit – ROM material would be dumped onto a grizzly screen and into the crusher dump hopper feeding a jaw crusher operating at an average utilization of 75% yielding an instantaneous design-throughput of 1,225 short tons per hour (st/h).
- Grinding Circuit – The grinding circuit incorporates a single semi-autogenous (SAG) mill, single ball mill design with an average utilization of 92%, yielding an instantaneous design-throughput of 998.5 st/h. When Historic Tailings are processed during early years of the operation, the slurry from the plant would also flow to the cyclone feed pump box. Cyclone underflow flows by gravity to the ball mill; cyclone overflow, at 33% solids with a target size of 80% passing (P_{80}) 75 microns, would be screened to remove tramp oversize and flow through a feed sample system and on to the antimony or gold rougher flotation circuit, depending on the antimony concentration of the material.
- Flotation Circuit (Antimony and Gold) – The flotation circuit consists of up to two sequential flotation stages to produce two different concentrates; the first stage of the circuit was designed to produce an antimony concentrate when the antimony grade is high enough, or bypassed if not, and the second stage was designed to produce a gold-rich concentrate.
- Pressure Oxidation Circuit – Two concentrate surge tanks would be pumped to the autoclave feed tank, which would feed the autoclave. The autoclave is designed to provide one hour of retention time at 428 degrees Fahrenheit to oxidize the sulfides and liberate the precious metals. Autoclave discharge would be processed through flash vessels and gas discharge is processed through a scrubber. Slurry discharge from the flash vessels would be processed through the basic ferric sulfate (BFS) re-leach tanks to stabilize the solids prior to cyanide leaching.
- Oxygen Plant – An oxygen plant producing 670 st/d of gas at 95 percent oxygen and a gauge pressure (psig) of 570 is planned. The oxygen would be from a vendor-owned oxygen plant located near the autoclave building providing the autoclave with an “over the fence” supply.
- Oxidized Concentrate Processing – Post-POX, the concentrate stream would be conditioned with lime and leached for 24 hours and discharged to a six stage pump-cell carbon-in-pulp (CIP) circuit for precious metal recovery from this high grade stream. The CIP tailings would be discharged to the flotation tailings leach circuit for extended retention time and to minimize reagent costs for the tailings leach system.
- Oxide Carbon-in-Leach and Tailings Detoxification – A (CIL) circuit was included in the design of the process plant to recover gold from non-refractory material in the flotation tailings, and in oxide material from the West End deposits that would be processed during oxidation circuit scheduled maintenance periods.
- Carbon Handling – Loaded carbon from the CIP circuit would be processed through a conventional carbon handling circuit.
- Gold Room – Precious metals would be recovered from the strip solution by electrowinning.
- Tailings – Tailings would be pumped from the process plant to the TSF In a HDPE-lined carbon steel pipe.
- Process Control Systems - The process plant design includes an integrated process control system.

1.19 PROJECT INFRASTRUCTURE

Site Access

The site is currently accessed by the Stibnite Road, National Forest (NF-412), from the village of Yellow Pine, with three alternative routes up to that point. To address a number of shortcomings related to these routes, alternative access via the Burntlog Route was selected over several other possible alternatives because it provides safer year-round access for mining operations, reducing the proximity of roads to streams, creeks and rivers, and this route respects the advice and privacy of community members close to the Project location.

Onsite and Offsite Facilities

In an effort to reduce traffic to and from the Project site and to reduce housing requirements at the site, administrative offices for Project would be located in or near the town of Cascade (the Cascade Complex). The Cascade Complex would include offices for some managers, safety and environmental services, human resources, purchasing, and accounting personnel. The Cascade Complex would also have a small warehouse, a parking area for trucks to check-in and assemble prior to traveling to the Project site and the main assay laboratory.

Midas Gold currently has an on-site facility capable of housing approximately 60 and feeding 125 workers per 12 hour shift. To manage the estimated peak construction workforce of 1,000-persons, the existing exploration camp would be relocated and expanded to provide the necessary accommodations. The operations camp would be developed by upgrading, and downsizing, the construction camp to meet the needs of the operations staff that would peak at over 500 persons.

Power Supply and Transmission

Grid power was selected as the best alternative for the electrical power supply for the Project based on its low operating cost and likely lowest environmental impact. In order provide the necessary power, the existing grid system would need to be upgraded to support the full anticipated 50 megawatt (MW) load of the Project. The upgrades would include an upgrade of approximately 42 mi of 69 kilovolt (kV) lines to 138 kV, new 138 kV substations at Lake Fork, Cascade, and Warm Lake, as well as measures to strengthen the voltages on the IPCo system. In addition, IPCo would re-supply small consumers between Warm Lake and Yellow Pine via a replacement 12.5 kV line. Construction power supply would be provided by three diesel generators that would then be used as emergency backup for the remainder of the operations of the Project.

Water Management and Supply

Water management infrastructure would be needed for surface water and sediment management and to provide water supply for both personnel and the operations. The PFS provides the framework for a comprehensive approach to water management at the Project site, addressing water management objectives for construction, operation, and post-closure. Key elements include segregation of process water, contact water, untreated stormwater, and sanitary waste from the environment, provision for fish passage around and then through the Yellow Pine pit during operations and after closure respectively, clean-up of legacy issues in the Project area, and reclamation and closure of the site to achieve acceptable and sustainable water quality.

Waste Management

Mine waste requiring on-site management includes waste rock from the three open pits, flotation and POX tailings from ore processing, and historic mine waste (spent heap leach ore from SODA and the Hecla heap, as well as historical waste dumps) exposed during construction and mining. The existing Historic Tailings would be reprocessed, and subsequently commingled with the rest of the tailings. A single TSF would be constructed to retain all tailings from the processing of the various ore types. The TSF would consist of a rockfill dam and a geosynthetic-lined impoundment that would be constructed in stages throughout the Project life. A majority of the waste rock would be deposited in the main WRSF located downstream of the TSF dam and would act as a buttress (enhancing dam stability), used as rockfill in TSF construction, or placed as backfill within mined-out areas of the pits to facilitate closure and reclamation. Current test work indicates no need for special handling of any of the waste materials. Spent ore and waste rock from previous on-site operations would be used as a construction material in the TSF. With SODA material included, the TSF dam and WRSF combined would hold 210 Mst of waste rock and overburden. Most of the waste from the West End pit would be used to backfill portions of the West End and Yellow Pine pits, with the remainder placed at the TSF and West End WRSF.

A geochemical characterization program was carried out for mine waste rock materials, including the spent ore on the SODA, which provides a basis for assessment of the potential for metal leaching and acid rock drainage, prediction of contact water quality, and evaluation of options for design, construction, and closure of the mine facilities. The results of the static geochemical test work demonstrate that the bulk of the Project waste rock material is likely to be net neutralizing and presents a low risk for acid generation, while there is still a potential to leach some constituents under the neutral to alkaline conditions (i.e. arsenic and antimony) both of which are currently elevated in ground and surface waters due to the naturally high geochemical background of these metals in the District and impacts from past mining activities. Similarly, bulk flotation tailings are expected to generate neutral pH drainage and require no special disposal considerations to prevent acidic drainage, and POX tailings will be blended with the bulk flotation tailings in order to benefit from their buffering capacity.

1.20 MARKET STUDIES AND CONTRACTS

The economic analysis completed for this PFS assumed that gold and silver production in the form of doré with payabilities, refining and transport charges as provided in Table 1.6.

Table 1.6: Doré Payables, Refining and Transportation Assumptions

Parameter	Gold in Doré	Silver in Doré
Metal Payability in Doré	99.5%	98.0%
Refining Charges	$1.00/oz Au	$0.50/oz Ag
Transportation Charges	$1.15/oz Au	$1.15/oz Ag

Table 1.7 summarizes the antimony concentrate payables and transportation charge assumptions for this PFS.

Table 1.7: Antimony Concentrate Payables and Transportation Assumptions

Parameter	Concentrate Payables and Transportation Charges
Antimony Payability	Constant at 68% (based on a constant life-of-mine concentrate grade of 59%)
Gold Payability	<5.0 g/t Au no payability ≥5.0 g/t ≤8.5 g/t Au payability of approximately 15 - 20% ≥8.5 g/t ≤10.0 g/t Au payability of approximately 20 - 25% ≥10.0 g/t Au payability of approximately 25%
Silver Payability	<300 g/t Ag no payability ≥300 g/t ≤700 g/t Ag payability of approximately 40 - 50% ≥700 g/t Ag payability of approximately 50%
Transportation Charges	$151/wet tonne from site to Asia

The metal prices selected for the four economic cases in this Report are shown in Table 1.8.

Table 1.8: Assumed Metal Prices by Case

Case	Metal Prices			Basis
	Gold ($/oz)	Silver(1) ($/oz)	Antimony(1) ($/lb)	
Case A	1,200	20.00	4.00	Lower-bound case that reflects the lower prices over the past 36 months and spot on December 1, 2014.
Case B (Base Case)	1,350	22.50	4.50	Approximate 24-month trailing average gold price as of December 1, 2014.
Case C	1,500	25.00	5.00	Approximate 48-month trailing average gold price as of December 1, 2014.
Case D	1,650	27.50	5.50	An upside case to show Project potential at metal prices approximately 20% higher than the base case.

Note:
(1) Prices were set at a constant gold:silver ratio ($/oz:$/oz) of 60:1 and a constant gold:antimony ratio ($/oz:$/lb) of 300:1 for simplicity of analysis, although individual price relationships may not be as directly correlated over time. Historic gold:silver ratios have averaged around 60:1.

1.21 ENVIRONMENTAL STUDIES

The Project area has been mined extensively for tungsten, antimony, mercury, gold, and silver since the early 1900s, providing strategic metals to the United States during war time critical minerals shortages, generating substantial economic benefit to the local counties and the State of Idaho, and providing much needed jobs and support to local businesses for nearly 100 years. These various historic mining efforts have left significant legacy environmental impacts that persist to this day, although multiple cleanup efforts undertaken by federal and state agencies and private entities have mitigated some of those historic impacts. Historic mining impacts have been compounded by extensive forest fires and subsequent damage from soil erosion, landslides and debris flows and resultant sediment transport.

In conjunction with the redevelopment of the Project area outlined in the PFS, Midas Gold has developed a plan to restore much of the site by removing existing barriers to fish migration and re-establishing salmon and steelhead fish passage, removing and reprocessing unconstrained historic tailings, reusing historic spent ore material for construction, restoring stream channels, and implementing sediment control projects such as repairing on Blowout Creek, as well as extensive reforestation of the Project area. Midas Gold has endeavored to minimize the Project's footprint and related impacts by siting facilities and roads on previously disturbed ground and away from riparian areas, provided for a new access road that avoids rivers and large waterways, and would connect to grid power to minimize fossil fuel consumption and haulage.

Baseline Studies and Existing Conditions

An extensive set of baseline data demonstrating historic and existing conditions exists for the Project site, including those collected by contractors for the US Forest Service (USFS) and the US Environmental Protection Agency (EPA) that determined there were no unacceptable risks to the environment or human health and that there were no populations (fish, wildlife, or human) shown as having a "likely" risk. In 2001, the EPA and the Bureau of Environmental Health and Safety, Division of Health, Idaho Department of Health and Welfare, determined the risk to be too low for listing on the National Priorities List. In 2009 and 2010, contractors to Midas Gold conducted Phase I and Phase II Environmental Site Assessments, as prescribed by ASTM International (ASTM) Standard Practices; these assessments determined that there were no imminent threats to human health or the environment, but that there was a number of pre-existing significant and moderate recognized environmental conditions.

In 2011, Midas Gold retained environmental consulting firms to conduct technical adequacy audits of all existing environmental information and to develop individual work plans to conduct an environmental baseline collection program. These workplans were developed with input from involved state and federal agencies in order to establish the existing environmental conditions, identify and quantify environmental risks and liabilities, and monitor for potential impacts from onsite activities. Work programs commenced in 2011 and will continue into 2015 and beyond to ensure an adequate baseline accurately describe the existing environment at the “brownfield site”, and allow for a "full and fair" discussion of all potentially significant environmental impacts in the event that the Stibnite Gold Project moves forward.

Consent Decrees

Several of the patented lode and mill site claims acquired by Midas Gold are subject to consent decrees entered in the US District Courts involving or pertaining to environmental liability and remediation responsibilities with respect to the affected properties, which provide regulatory agencies access and the right to conduct remediation activities and also require that heirs, successors and assignees refrain from activities that would interfere with or adversely affect the integrity of any remedial measures implemented by government agencies.

Permitting

Should a decision be made to file a Plan of Operations (PoO), approval of any Final PoO / Reclamation Plan for the Project would require an environmental analysis in compliance with the National Environmental Policy Act (NEPA), which requires federal agencies to study and consider the likely environmental impacts of the proposed action before taking whatever federal action is necessary for the Project to proceed. An EIS serves as an "overarching" permit requirement, as well as that for water discharge; waste and tailings placement and endangered species authorization. The EIS Record of Decision (ROD) effectively drives the entire permitting process, since a favorable ROD is required before these important clearances can be obtained. State and local permitting processes would be integrated and proceed concurrent with the EIS, and include air quality, cyanide, land application of water, groundwater, water rights, dam safety, reclamation, building permits, sewer and water systems, etc. Midas Gold believes it will be beneficial to have all permit processes integrated into the Idaho Joint Review Process (IJRP) and that the IJRP would play a key role in increased communication and cooperation between the various involved governmental agencies, and reduced conflict, delay, and costs in the permitting process. Midas Gold’s objective is to make the Project a fully integrated, sustainable, and socially and environmentally responsible operation through open communications and accessibility.

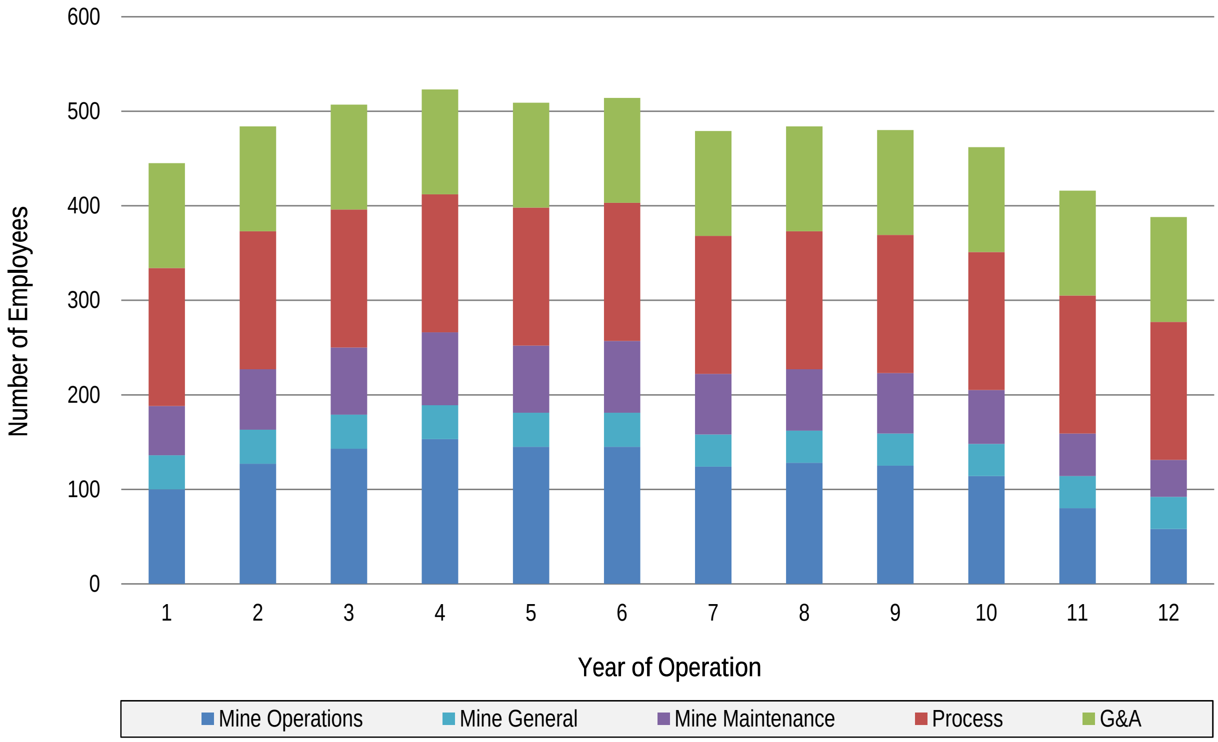
1.22 SOCIAL IMPACTS

Employment

Populations continue to grow in Valley and Adams Counties, but jobs are not keeping pace; unemployment rates in these counties are some of the highest in Idaho, while wages average only $27,433/year. The Project could do much to improve this situation, with current mining jobs in Idaho averaging $72,500/year and the Project offering an approximate average of some 400 direct and 321 indirect and induced jobs in Idaho generating aggregate annual payrolls of $48 million/year during the 3-year construction period (plus additional out-of-state contractors for specialized construction functions) and an approximate average of some 500 direct and 439 indirect and induced jobs generating aggregate annual payrolls of $56 million/year during the 12-year operating period.

Operations are scheduled for 365 days/year; a breakdown of the annual staffing requirements to operate and maintain the mine, processing plant, and appurtenant facilities and functions for the five functional work areas is provided on Figure 1.7. Whenever possible, the work force was segregated between the mine site and the Cascade Complex to limit the number of personnel at the mine site that require residential support and transportation to and from site.

Figure 1.7: Annual Direct Employment by Department



Taxes

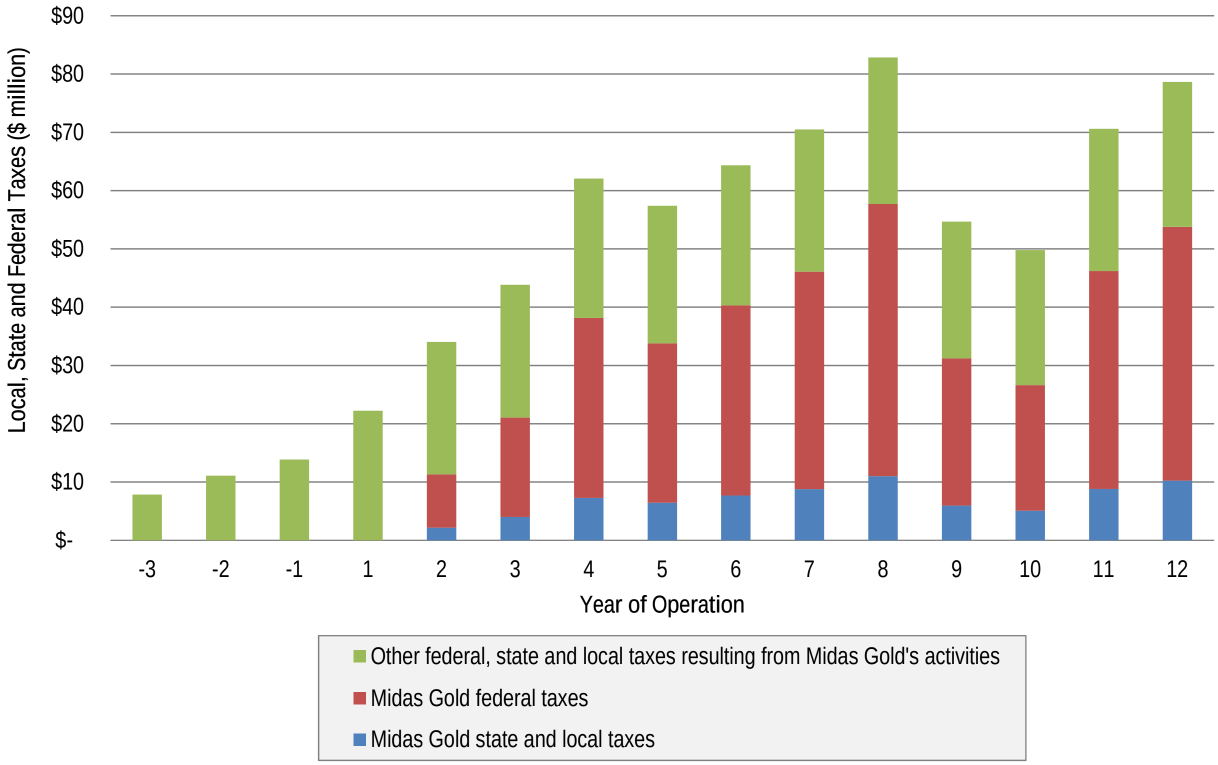
To estimate the potential economic impacts from the Project, an economic impact model known as IMpact analysis for PLANning (IMPLAN) was constructed (Peterson, 2014). The IMPLAN model was used to estimate direct, indirect and induced taxes, that would be paid by other taxpayers (other than Midas Gold), and the tax estimates were combined with the direct federal, state and local taxes that would be paid by Midas Gold (see Section 22 for details on the PFS financial model and tax calculations) to develop an estimate for the overall taxes generated by the Project. Figure 1.8 presents a plot of estimated annual direct, indirect and induced taxes associated with the Project paid by both Midas Gold and other taxpayers to federal, state and local governments.

Taxes that would be paid directly by Midas Gold over the life of the Project, based on the assumptions in the PFS, are estimated at approximately $329 million in federal corporate income taxes, and $86 million in state corporate income and mine license taxes.

Additional indirect and induced taxes that result from Midas Gold's activities that would be paid by other taxpayers, based on the assumptions in the PFS, are estimated at approximately $177 million in federal taxes (including payroll, excise, income and corporate), and $131 million in state and local taxes (including property, sales, excise, personal, corporate, and other).

Total direct, indirect and induced taxes are therefore estimated at $506 million in federal taxes and $218 million in state and local taxes, representing a significant contribution to the economy during the 15 year construction and operating life of the Project.

Figure 1.8: Chart of Estimated State and Federal Taxes



Environmental Mitigation and Remediation

Midas Gold has made considerable effort to design the Project restoration of the site through the incorporation of specific mitigation and remediation components, including re-establishing fish passage, removal and reprocessing of unconstrained Historic Tailings, removal of unconstrained historical waste rock, reuse of historical spent ore piles for construction, stream channel restoration projects, and sediment control. The mitigation and remediation activities and costs are summarized in Section 20 and Section 21, respectively. Additionally, the Project design team has optimized siting of facilities wherever possible to avoid riparian areas, limit stream crossings, position facilities on previously disturbed ground, move major access routes away from large waterways, minimize the number of people on site to limit traffic, and re-establish historic line power to the site to minimize fuel haulage and reduce greenhouse gas emissions. In some cases, disturbance of albeit already impacted wetlands and streams would be unavoidable, which disturbance Midas Gold intends to address through a mitigation bank or similar entity as well as through onsite replacement and restoration of existing wetlands. Midas Gold would continue to build on its strong record by continuing to proactively evaluate Best Management Practices (BMPs) and Standard Operating Procedures (SOPs) effectiveness, including a post-closure component.

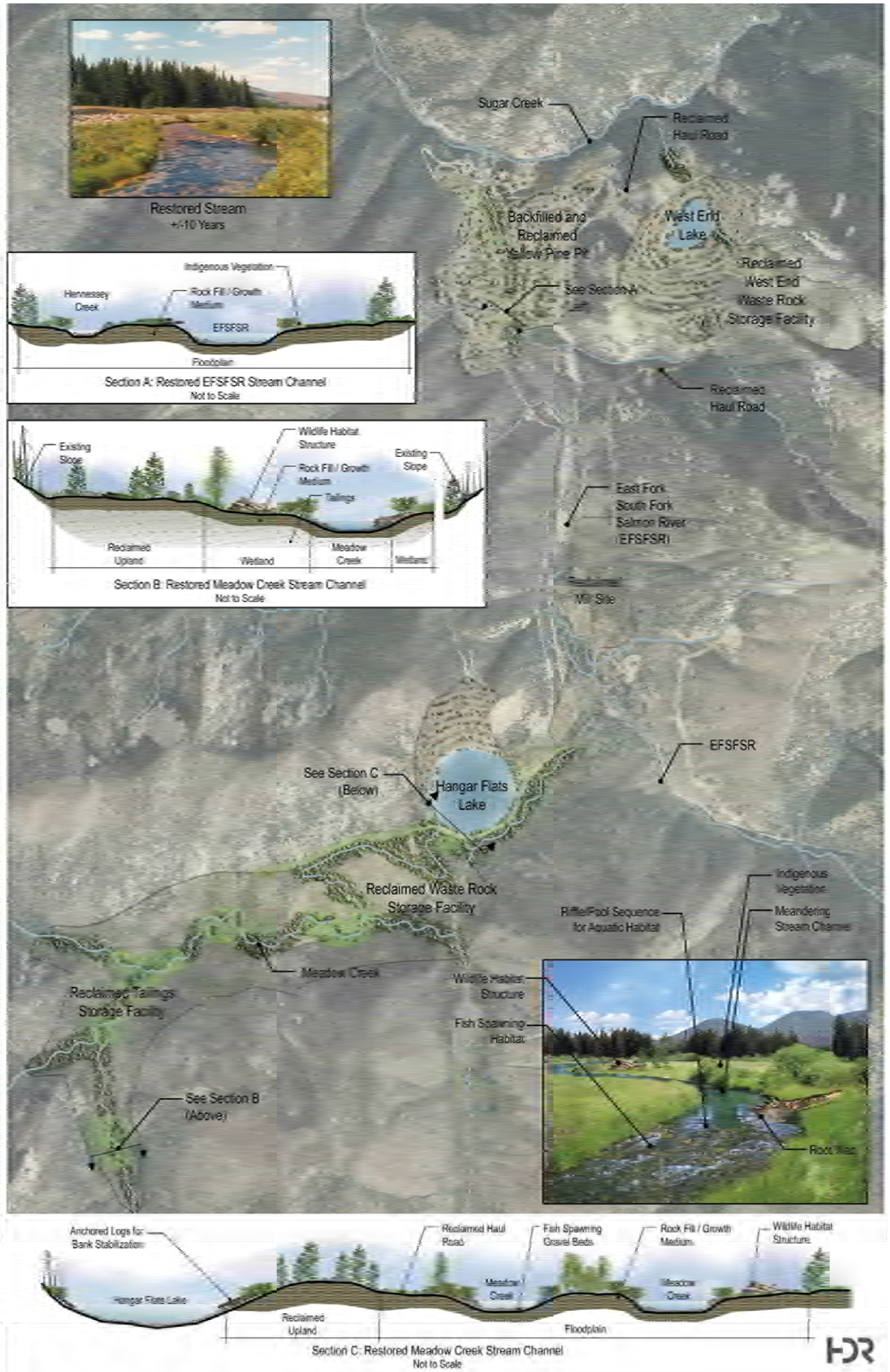
A critical goal for Midas Gold has been the incorporation of fisheries protection and habitat restoration components aimed at achieving a sustainable anadromous fishery, including passage of migrating salmon, steelhead, and trout to the headwaters of the EFSFSR both during and after operations for the first time since 1938. Upon closure, new enhanced wetlands and spawning grounds would be established to assist in the return of fish migration and reestablishment of a health riparian zone along the rebuilt stream channel. Midas Gold has also incorporated efforts to improve water quality by removing historical tailings, spent ore and waste rock and respectively reprocessing, reusing and relocating these materials, as well as developing sediment control features for Blowout Creek, currently a major contributor of sediment, and replanting historically disturbed and forest fire affected areas to reduce sedimentation.

Closure

During construction, operations and once operations cease, extensive reclamation would be completed, creating enhanced surface water systems and suitable fisheries habitat. Midas Gold has identified 17 priority Project conservation components that form the basis of the overall conservation strategy that are summarized in Section 20; Figure 1.9 presents a site-wide illustration of the overall closure strategy. These components include: construction of the new Burntlog Road (which effectively moves the primary transportation route away from the Johnson Creek fishery), backfilling the Yellow Pine pit with environmentally appropriate material to create a stable hydrogeologic gradient suitable to the current conditions, closure of historic mine workings on USFS lands, ongoing wetlands and stream habitat enhancement, permanent restoration of fish passage up the EFSFSR, post-closure wetlands and stream habitat enhancement on top of the Meadow Creek TSF surface and reforestation of the Project area. The conservation commitment to restore the site through implementation of these measures is discussed in greater detail in Section 20, while closure costs are detailed in Section 21.

When operations cease, mobile and salvageable equipment would be removed, and foundations broken up, covered and re-vegetated (Figure 1.9). The objective is for the development of a self-sustaining natural environment that has addressed many of the historical impacts and supports a healthy fish and wildlife population. Post-closure monitoring is planned for an extended period to ensure that these objectives have been met.

Figure 1.9: Conceptual Post Closure Reclamation



1.23 ECONOMIC ANALYSIS

Capital and operating cost estimates were developed based on Q3 2014, un-escalated U.S. dollars. Vendor quotes were obtained for all major equipment. Most costs were developed by first principles, although some were estimated based on factored references and experience with similar projects.

Capital Costs

The estimated capital expenditure or capital costs (CAPEX) for the Project consists of four components: (1) the initial CAPEX to design, permit, pre-strip, construct, and commission the mine, plant facilities, ancillary facilities, utilities, operations camp, and on and off site environmental mitigation; (2) the sustaining CAPEX for facilities expansions, mining equipment replacements, expected replacements of process equipment and ongoing environmental mitigation activities during the operating period; (3) working capital to cover delays in the receipts from sales and payments for accounts payable and financial resources tied up in inventory, and (4) closure CAPEX to cover post operations reclamation costs. Initial and working CAPEX are the two main categories that need to be available to construct the Project. Table 1.9 summarizes the initial, sustaining and closure CAPEX for the Project.

Table 1.9: Capital Cost Summary

Area	Detail	Initial CAPEX ($000s)	Sustaining CAPEX ($000s)(2)	Closure CAPEX ($000s)(2)	Total CAPEX ($000s)
Direct Costs	Mine Costs	47,552(1)	35,346	-	82,898
	Processing Plant	336,219	1,579	-	337,798
	On-Site Infrastructure	149,245	39,937	-	189,182
	Off-Site Infrastructure	80,327	-	-	80,327
Indirect Costs		176,687	4,275	-	180,962
Owner's Costs		26,806	-	-	26,806
Environmental Mitigation Costs		10,606	8,165	-	18,771
Closure Bonding, Closure and Reclamation Costs		762	9,185	56,542	66,489
Total CAPEX without Contingency		828,204	98,488	56,542	983,233
Contingency		142,050	-	-	142,050
Total CAPEX with Contingency		970,254	98,488	56,542	1,125,283

Note:
(1) Initial mining CAPEX includes environmental remediation costs as discussed in Section 21.
(2) Contingency included in line items.

Mitigation costs only refer to relocation of a certain portion of the readily identifiable and quantified waste from historical mining activities; other costs related to recovery and reprocessing of Historic Tailings and relocation of unquantified waste rock at West End and Yellow Pine are included in operating costs and are partially offset by recovery of gold and antimony from the Historical Tailings.

Operating and All-In Costs

The cash operating costs include mine operating costs, process plant operating costs, general and administrative (G&A) costs, while total cash costs include smelting and refining charges, transportation charges, and royalties. A detailed breakdown of the summary of the operating costs (OPEX) costs is presented in Table 1.10. The details that comprise the OPEX are provided Section 21. The All-In Sustaining Costs (AISC) are also provided in the table, as well as the All-In Costs (AIC), which include non-sustaining capital and closure and reclamation costs.

Table 1.10: Operating Cost, AISC and AIC Summary

Total Production Cost Item	LOM			Years 1-4	
	($/st mined)	($/st milled)	($/oz Au)	($/st milled)	($/oz Au)
Mining	2.00	9.08	222	10.04	222
Processing	-	14.45	354	14.10	312
G&A	-	3.13	77	3.01	67
Cash Costs Before By-Product Credits	-	26.65	653	27.15	601
By-Product Credits	-	-3.45	-85	-5.32	-118
Cash Costs After of By-Product Credits	-	23.20	568	21.83	483
Royalties	-	0.94	23	0.34	23
Refining and Transportation	-	0.25	6	1.04	8
Total Cash Costs	-	24.38	597	23.20	513
Sustaining CAPEX	-	1.00	24	0.52	11
Salvage	-	-0.27	-7	0.00	0
Property Taxes	-	0.04	1	0.04	1
All-In Sustaining Costs	-	25.15	616	23.76	526
Reclamation and Closure(1)	-	0.58	14	-	-
Initial (non-sustaining) CAPEX(2)	-	9.89	242	-	-
All-In Costs	-	35.62	872	-	-

Notes:
(1) Defined as non-sustaining reclamation and closure costs in the post-operations period.
(2) Initial Capital includes capitalized preproduction.

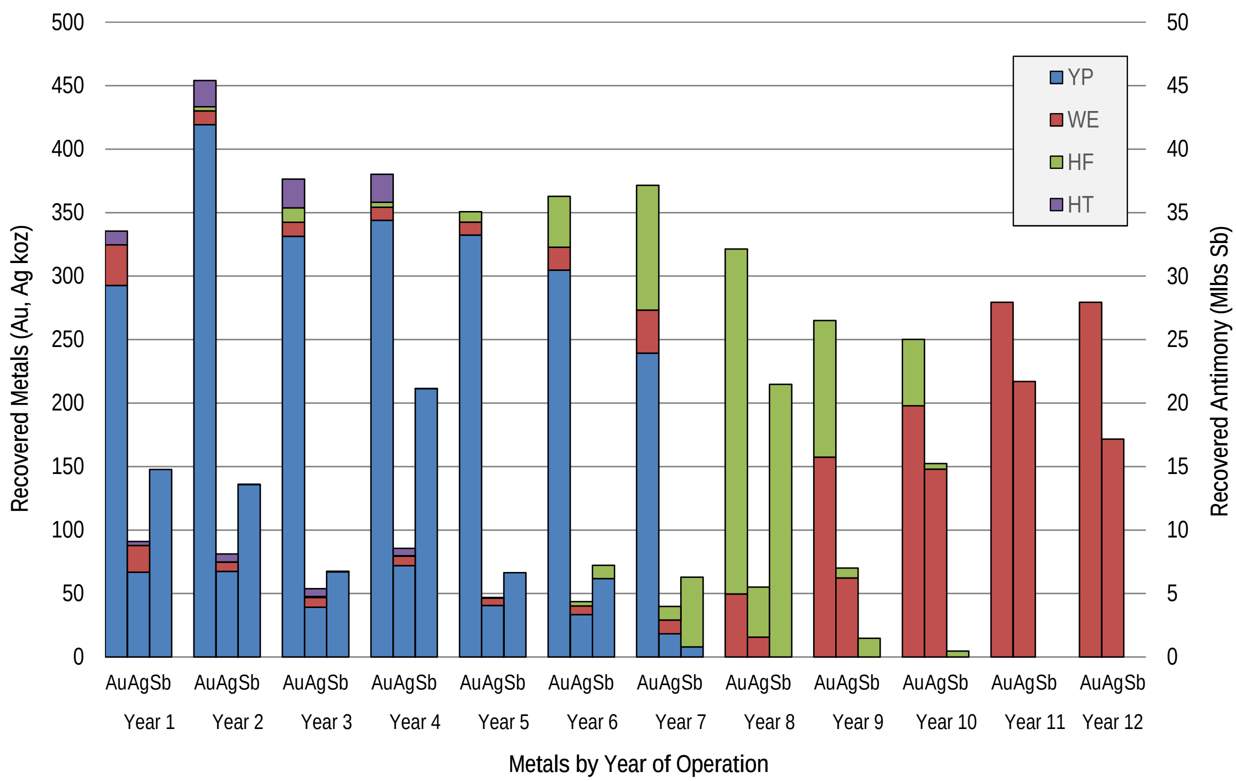
Metal Production

Recovered metal production by deposit is summarized in Table 1.11 and illustrated on an annual basis on Figure 1.10.

Table 1.11: Recovered Metal Production

Product by Deposit	Gold (koz)	Silver (koz)	Antimony (klbs)
Doré Bullion			
Yellow Pine	2,263	338	-
Hangar Flats	597	68	-
West End	1,090	681	-
Historic Tailings	72	20	-
Doré Bullion Recovered Metal Totals	4,023	1,107	-
Antimony Concentrate			
Yellow Pine	12	611	69,822
Hangar Flats	5	349	30,030
Antimony Concentrate Recovered Metal Totals	17	960	99,852
Total Recovered Metals	4,040	2,067	99,852

Figure 1.10: Annual Recovered Metals by Deposit



Economic Analysis

The economic model described herein is not a true cash flow model as defined by financial accounting standards but rather, a representation of Project economics at a level of detail appropriate for a PFS level of engineering and design. The first year of analysis starts with the decision point of the Project, the completion of the EIS, and preliminary permit approval (Year -3 or three years before the start of commercial production). Taxation was taken into account using current federal, state, and county rates but the overall tax calculation is approximate and uses rudimentary depletion and depreciation estimates.

Four cases were run in the economic model to present a range of economic outcomes using varying metal prices. The metal prices used in the economic model are shown in Table 1.8 and off-site costs and payables used are in Table 1.6 and Table 1.7. There is no guarantee that any of the metal prices used in the four cases are representative of future metals prices. The constant parameters for all cases are shown in Table 1.12.

Table 1.12: Economic Assumptions used in the Economic Analyses (all Cases)

Item	Unit	Value
Net Present Value Discount Rate	%	5
Federal Income Tax Rate	%	35
Idaho Income Tax Rate	%	7.4
Idaho Mine License Tax	%	1.0
Valley County Rural Property Tax Rate ($/$1,000 market value)	%	0.063
Percentage Depletion Rate for Gold and Silver	%	15
Percentage Depletion Rate for Antimony	%	22

Item	Unit	Value
Depreciation Term	Years	7
Equity Finance	%	100
Capital Contingency (Overall)	%	17.2

The results of the economic analyses are shown in Table 1.13. Based on the assumptions made in this PFS, the $ATNPV_{5\%}$ is estimated to be $832 million yielding an after-tax IRR of 19.3%. The $ATNPV_{5\%}$ and IRR increase considerably with the Case C metal prices and decreases with the Case A metal prices. The $PTNPV_{5\%}$ for Case B was estimated to be $1,093 million with an IRR of 22.0%.

Table 1.13: Economic Results by Case

Parameter	Unit	Pre-tax Results	After-tax Results
Case A ($1,200/oz Au, $20.00/oz Ag, $4.00/lb Sb)			
$NPV_{0\%}$	M$	1,286	1,041
$NPV_{5\%}$	M$	662	513
IRR	%	16.2	14.4
Payback Period	Production Years	4.0	4.1
Case B ($1,350/oz Au, $22.50/oz Ag, $4.50/lb Sb)			
$NPV_{0\%}$	M$	1,915	1,499
$NPV_{5\%}$	M$	1,093	832
IRR	%	22.0	19.3
Payback Period	Production Years	3.2	3.4
Case C ($1,500/oz Au, $25.00/oz Ag, $5.00/lb Sb)			
$NPV_{0\%}$	M$	2,543	1,929
$NPV_{5\%}$	M$	1,524	1,129
IRR	%	27.2	23.4
Payback Period	Production Years	2.6	2.9
Case D ($1,650/oz Au, $27.50/oz Ag, $5.50/lb Sb)			
$NPV_{0\%}$	M$	3,171	2,344
$NPV_{5\%}$	M$	1,955	1,414
IRR	%	31.9	27.0
Payback Period	Production Years	2.2	2.5

The contribution to the Project economics, by metal, is about 94% from gold, 5% from antimony, and less than 1% from silver. The undiscounted after-tax cash flow for Case B is presented in Figure 1.11. The payable metal value by year for Case B is summarized on Figure 1.12.

Figure 1.11: Undiscounted After-Tax Cash Flow for Base Case B

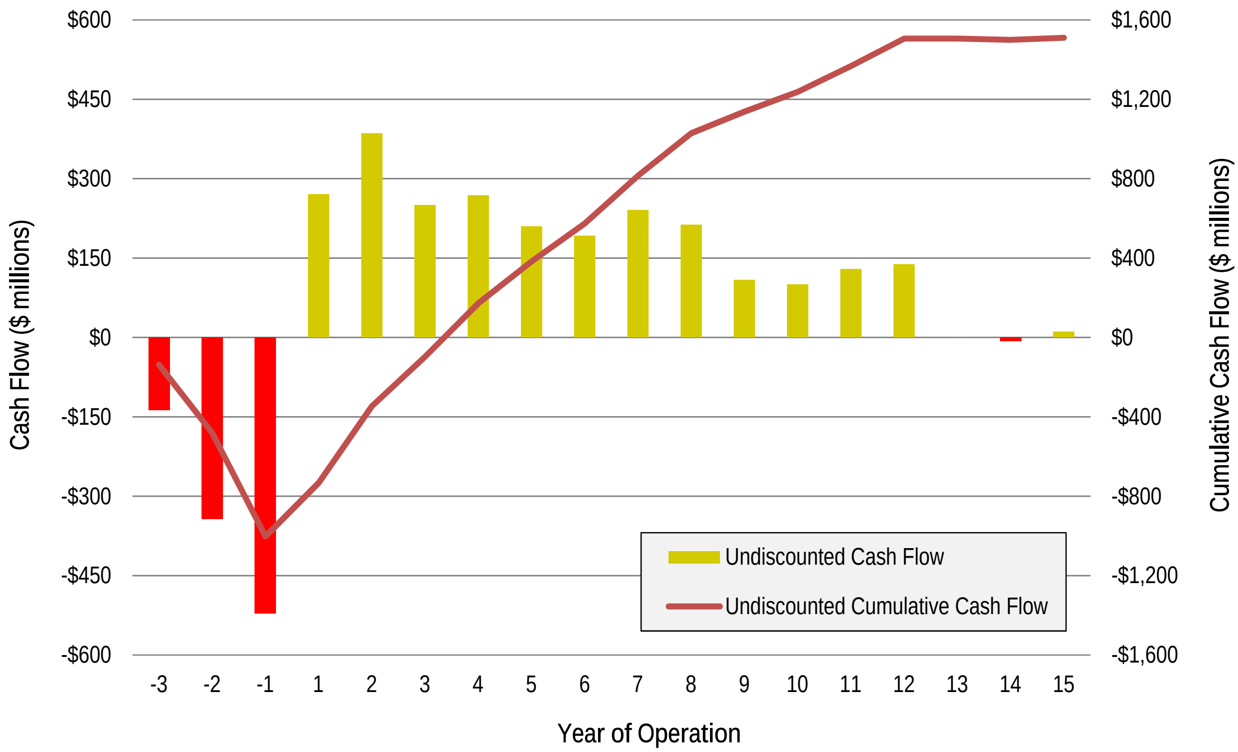
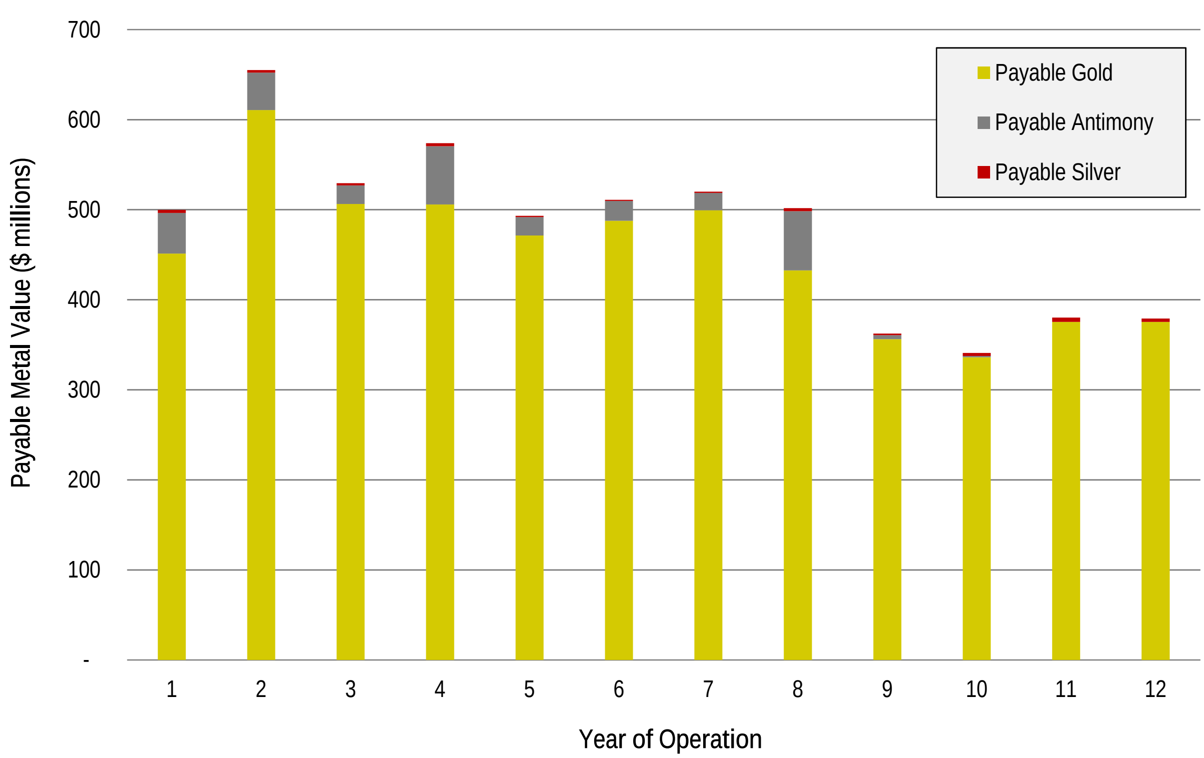


Figure 1.12: Payable Metal Value by Year for Case B



Mine Life

Using the current Mineral Reserve and the nominal design throughput of 22,050 st/d, the mine plan projects a 12 year production life. Construction is projected to require a three-year period after the permits are obtained and prior to the start of operations. Closure is projected to take at least 10 years post-production, with some reclamation work occurring concurrently with operations, and the bulk of the closure activities and costs incurred in the first 3 years after operations cease. Some closure activities and long-term monitoring are anticipated to continue well after the reclamation period is complete to ensure that the closure designs continue to protect the environment and are performing in accordance with the design parameters.

Sensitivity Analysis

Sensitivity analyses were performed using metal prices, mill head grade, CAPEX, and OPEX as variables. The value of each variable was changed plus and minus 20% independently while all other variables were held constant. The results of the sensitivity analyses are shown in Table 1.14 and Table 1.15.

Table 1.14: Pre-tax $NPV_{5\%}$ Sensitivities by Case

Case	Variable	$PTNPV_{5\%}$ (M$)		
		-20% Variance	0% Variance	20% Variance
Case A	CAPEX	862	662	463
	OPEX	1,017	662	308
	Metal Price or Grade	-27	662	1,352
Case B (Base Case)	CAPEX	1,292	1,093	894
	OPEX	1,447	1,093	739
	Metal Price or Grade	318	1,093	1,869
Case C	CAPEX	1,723	1,524	1,325
	OPEX	1,878	1,524	1,170
	Metal Price or Grade	662	1,524	2,386
Case D	CAPEX	2,154	1,955	1,755
	OPEX	2,309	1,955	1,600
	Metal Price or Grade	1,007	1,955	2,902

Table 1.15: After-tax $NPV_{5\%}$ Sensitivities by Case

Case	Variable	$ATNPV_{5\%}$ (M$)		
		-20% Variance	0% Variance	20% Variance
Case A	CAPEX	676	513	346
	OPEX	760	513	239
	Metal Price or Grade	-30	513	1,012
Case B (Base Case)	CAPEX	980	832	674
	OPEX	1,057	832	577
	Metal Price or Grade	244	832	1,357
Case C	CAPEX	1,266	1,129	982
	OPEX	1,341	1,129	903
	Metal Price or Grade	513	1,129	1,696
Case D	CAPEX	1,548	1,414	1,277
	OPEX	1,623	1,414	1,200
	Metal Price or Grade	770	1,414	2,035

1.24 RISKS AND OPPORTUNITIES

A number of risks and opportunities have been identified in respect of the Project; aside from industry-wide risks and opportunities (such as changes in capital and operating costs related to inputs like steel and fuel, metal prices, permitting timelines, etc.), high impact Project specific risks and opportunities are summarized below.

Risks, for which additional information is required in order to mitigate:

- Use of historical data in Mineral Resource estimates, which could affect these estimates;
- Limited geotechnical data which could change pit slopes or foundation conditions in infrastructure areas;
- Loss of gold into antimony concentrates;
- Water management and chemistry, which could affect diversion and closure designs and/or the need for long term water treatment; and
- Construction schedule.

Opportunities that could improve the economics, and/or permitting schedule of the Project, including a number with potential to increase the $NPV_{5\%}$ by more than $100 million follow:

- In pit conversion of Inferred Mineral Resources to Mineral Reserves, increasing Mineral Reserves and reducing strip ratio;
- Out of pit conversion of Inferred Mineral Resources to Mineral Reserves adjacent to the current Mineral Reserves, resulting in increased Mineral Reserves in close proximity to planned pits;
- In pit conversion of unclassified material currently treated as waste rock to Mineral Reserves, increasing Mineral Reserves and reducing strip ratios;
- Improved continuity of higher grade gold mineralization in the Yellow Pine pit, particularly around the area with excluded or limited Bradley drilling, increasing grade of the Mineral Reserves;

- Additional fire assay information at West End in areas where only cyanide assays were available, potentially increasing grade and Mineral Reserves;
- Potential additional antimony mineralization and/or grade in areas where Bradley data was eliminated and/or areas where antimony was not assayed, increasing by-product credits;
- Potential for the definition of a higher grade, higher margin underground Mineral Reserve at Scout and Garnet; and
- Discovery of other new deposits with attractive operating margins.

Opportunities with a medium impact ($10 to $100 million increase in Project $NPV_{5\%}$) include improved recoveries, secondary processing of antimony concentrates, potential legislative designation of antimony as a critical mineral; steeper pit slopes, onsite quicklime generation, and government funding of off-site infrastructure. A number of lesser impact opportunities also exist.

1.25 CONCLUSIONS AND RECOMMENDATIONS

Industry standard mining, processing, construction methods, and economic evaluation practices were used to assess the Project. There was adequate geological and other pertinent data available to generate the PFS.

The financial analysis presented in Section 22 of the PFS demonstrates that the Project is financially viable and has the potential to generate positive economic returns based on the assumptions and conditions set out in this Report, while other sections of the PFS demonstrate that the Project is technically and environmentally viable. These conclusions warrants continued work to advance the Project to the next level of study, which is a Feasibility Study (FS), by conducting the work indicated in the recommendations section of this Report. These recommendations form a single phase that will move the Project through to completion of a FS and, if so desired, through the regulatory process for mine development. Total estimated costs for completion of this single phase are $22.3 million. While additional information is required for a complete assessment of the Project, at this point there do not appear to be any fatal flaws. The PFS has achieved its original objective of providing a review of the potential economic viability of the Project to standards appropriate for a PFS.

The QPs of this Report are not aware of any unusual, significant risks or uncertainties that could be expected to affect the reliability or confidence in the Project based on the data and information available to date.

An additional $22.5 million is identified as discretionary expenditures that would target a number of the opportunities identified in Section 25 of this PFS Report that could enhance the PFS case but that are not required in order to complete a FS or permitting.

Table 1.16: Project Development Work Program Budget

Recommendations and Work Program	Estimated Costs ($000s)	
	Core	Discretionary
Mineral Resource Evaluation and Exploration	3,700	21,200
Field Programs Required for FS	1,900	-
Metallurgical Testing Required for FS	2,400	1,300
FS-Level Engineering	3,500	-
Environmental, Regulatory Affairs and Compliance	10,800	-
Totals	22,300	22,500

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2 INTRODUCTION

This prefeasibility study technical report (PFS or Report) was commissioned by Midas Gold, for its Stibnite Gold gold-antimony-silver Project (**Stibnite Gold Project** or **Project**) at Stibnite, Idaho. This Report has been prepared for Midas Gold Corp. (**MGC**), a British Columbia company that owns and operates the Project through its wholly-owned subsidiaries, Midas Gold, Inc. (**MGI**), MGI Acquisition Corp (**MGIAC**), Idaho Gold Holding Company (**IGHC**) and Idaho Gold Resources, LLC (**IGR**). Unless the context indicates otherwise, references throughout this Report to "**Midas Gold**" includes one or more of the aforementioned subsidiaries of MGC.

The Report has been prepared in compliance with the Canadian Securities Administrators (**CSA**) National Instrument 43-101 (**NI 43-101**) standards for reporting mineral properties, Companion Policy 43-101CP, and Form 43-101F1. The contents of this Report reflect the technical and economic conditions at the effective date of the Report. These conditions may change significantly over time; consequently, actual results may vary considerably from those depicted herein.

2.1 PURPOSE OF REPORT

This Report provides a comprehensive overview of the Project and includes recommendations for future work programs required to advance the Project to a decision point. This Report defines an economically feasible, technically and environmentally sound Project that minimizes impacts and maximizes benefits. The Stibnite Gold Project key considerations are as follows:

- The Project design began with the end in mind, contemplating the development, operation and closure of the Project on a sustainable basis, meeting the needs of the present and enhancing the ability of future generations to meet their own needs. The Project design incorporates the key concepts of meeting the needs of society for a better life, providing economic prosperity and remaining protective of the environment.
- The Project is designed to ensure ongoing positive local and regional fiscal and social benefits through taxation, employment, and business opportunities in a region where the economy has suffered for more than a decade, resulting in some of the highest unemployment and lowest annual wages in Idaho.
- From the beginning, the Project has been designed for what will remain after closure. The plan for closure is protective of the environment and incorporates inherently stable, secure features that will provide the foundation for an evolution through time to a naturally sustainable ecosystem.
- The Project design incorporates the repair of extensive historical mining-related impacts much of which would occur during initial construction and early operations.
- The new facilities contemplated for the Project are tightly constrained and, to a large extent, placed in historically impacted areas in order to minimize the incremental Project footprint.
- Salmon and other fishery enhancement is integral to the Project design. Removal of man-made barriers and reconstruction of natural habitat would allow salmon and other fish migration into the upper reaches of the watershed for the first time since 1938.
- During development, operations and closure, all aspects of the Project are designed to improve existing conditions where possible and remain protective of the environment, with the extensive costs related to remediation and reclamation of historical impacts accommodated by an economically feasible Project.

This Report provides information about the geology, mineralization, exploration, mineral resource potential, mining method, process method, infrastructure, social and economic benefits, environmental protection, repair of historical impacts, reclamation and closure concepts, capital and operating costs and an economic analysis for the Project. Economic and technical analyses included in this Report provide only a summary of the potential Project economics based on the many assumptions set out herein. There is no guarantee that the Project economics described herein can be achieved.

This Report and the information contained herein is current as of the effective date of the Report and supersedes earlier technical reports completed for Midas Gold including the Preliminary Economic Assessment (PEA) Technical Report for the Golden Meadows Project Idaho, dated September 21, 2012 (SRK, 2012); the information in the PEA should no longer be relied upon.

2.2 SOURCES OF INFORMATION AND QUALIFIED PERSONS

The sources of information include data and reports supplied by Midas Gold personnel, and documents referenced in Section 27. M3 used its experience to determine if the information from previous reports was suitable for inclusion in this Report and adjusted information that required amending. Revisions to previous data were based on research, recalculations and information from other projects. The level of detail utilized was appropriate for this level of study.

This PFS is based on information collected by the Qualified Persons (each a QP) during their site visits. In addition, a number of meetings were conducted between M3 and Midas Gold. This Prefeasibility Study Report is based on the following sources of information.

- Personal inspection of the Stibnite Gold Project site and surrounding area.
- Technical information provided to the QPs by Midas Gold through various reports.
- Budgetary quotes from vendors for engineered equipment.
- Technical and cost information provided by Idaho Power Co. to Power Engineers concerning power supply for the Project.
- Technical and economic information subsequently developed by M3 and associated consultants.
- Information provided by other experts with specific knowledge and expertise in their fields as described in Section 3 of this Report, Reliance on Other Experts.
- Additional information obtained from public domain sources.
- The information contained in this Report is based on documentation believed to be reliable. Information utilized in this Report will be either retained in Midas Gold's offices in Boise, Idaho or readily available from Midas Gold's consultants' Project files, subject to an appropriate agreement concerning confidentiality.

The individuals who have provided input to this PFS have extensive experience in the mining industry and are members in good standing of appropriate professional institutions. Table 2.1 provides a list of the QPs, their affiliation, sections for which they are responsible, date of the most recent site visit, and items reviewed on their site visits. The QP Certificates are provided as Appendix I.

Table 2.1: List of Qualified Persons

Qualified Person	Company	Section Responsibility	Site Visit Date	Site Visit Review
Lee A. Becker, P.E.	M3 Engineering & Technology Corp.	18 (except 18.2, 18.9, 18.10, 18.11, and 18.12), 21 (except 21.1.2 and 21.2.3), 22, 24, 25, 26	July 18, 2017	General site visit.
Richard K. Zimmerman, P.G., SME-RM	M3 Engineering & Technology Corp.	1, 2, 3, 4, 5, 6, 19, 23, 24, 25, 26, 27	March 7, 2013	General site visit.
Garth D. Kirkham, P. Geo	Kirkham Geosystems Ltd.	7, 8, 9, 10, 11, 12, 14	April 23-25, 2014, and July 14-15, 2014	On both visits toured core logging and storage facilities. Site visit entailed inspection of the shops, offices, reclaimed drill sites, the Yellow Pine, Hanger Flats and West End mineral resource areas along with the outcrops, historic drill collars and areas of potential disturbance for potential future mining operations.
Christopher J. Martin, C.Eng.	Blue Coast Metallurgy Ltd.	13	August 25, 2011	General site visit.
John M. Marek, P.E.	Independent Mining Consultants Inc.	15, 16, 21.1.2, 21.2.3	September 16-17, 2013	Reviewed Project geology, terrain, and operational constraints at site. Visited drill core handling facility to review logging, sampling, and handling procedures.
Allen R. Anderson, P.E.	Allen R. Anderson Metallurgical Engineer Inc.	17	1	
Richard C. Kinder, P.E.	HDR Engineering Inc.	18.2	October 12, 2012	Route survey, where practicable, of access road options evaluated.
Peter E. Kowalewski, P.E.	Tierra Group International Ltd.	18.9, 18.10, 18.11, 18.12, 20	March 7, 2013	General site visit.

Notes:
1) Allen Anderson has not visited the site.

2.3 ABBREVIATIONS, UNITS, AND TERMS OF REFERENCE

This PFS is intended for the use of Midas Gold for the further advancement of the Stibnite Gold Project toward the feasibility study phase. It provides a mineral resource estimate, a classification of mineral resources in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) classification system and an evaluation of the Project, which presents a current view of the potential economic outcome.

Imperial units (American System) of measurement are used in this Report. Other units of measurement used in this Report are defined when first used. Abbreviations are given in Section 2.3.4. All monetary values are in U.S. dollars ($) unless otherwise noted.

2.3.1 Mineral Resources

As required by NI43-101, the Mineral Resources and Mineral Reserves in this Report have been classified according to the "*CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines*" (May 2014). Accordingly, the Mineral Resources have been classified as Measured, Indicated or Inferred, the Mineral Reserves have been classified as Proven, and Probable based on the Measured and Indicated Mineral Resources as defined below.

> *"A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge."*

> *"An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes."*

> *"An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed."*

2.3.2 Mineral Reserves

As required by NI43-101, Mineral Reserves have been defined according to the "*CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines*" (May 2014):

> *"A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.*

> *A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified."*

2.3.3 Glossary

Table 2.2 provides a glossary of certain terms that are used in this Report.

Table 2.2: Glossary

Term	Definition
Albion	A patented metallurgical process developed by Glencore that involves recovering metals using a combination of ultrafine grinding and oxidative leaching at atmospheric pressure.
Assay	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure	All expenditures not classified as operating costs, but excluding corporate sunken costs such as acquisition.
Cascade Complex	administrative offices for Project located in or near the town of Cascade offices for managers, safety and environmental services, human resources, purchasing, and accounting personnel
Composite	Combining more than one sample result to give an average result over a larger distance.
Concentrate	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing	Initial process of reducing ore particle size by impact to render it more amenable for further processing.
Cut-off Grade (CoG)	The grade of mineralized rock above which it becomes profitable to extract the mineralization.
Dilution	Waste, which is rock below an economic cutoff value mined with ore.
Dike	A sheet of igneous rock intruded along a crack in a rock mass and crystallized in place.
Dip	Angle of inclination of a geological feature/rock from the horizontal.
District	A bounded division and organization of a mining region.
Fault	The surface of a fracture along which movement has occurred.
Gangue	Non-valuable components of the ore.
Grade	The measure of concentration of a specific mineral within mineralized rock.
Historic Tailings	Approximately 3 Mt of uncontained tailings deposited in the Meadow Creek Valley by previous operators.
Hydrocyclone	A process whereby particulate materials are segregated by size by exploiting the interaction between gravitational and centrifugal forces.
Igneous	Primary crystalline rock formed by the solidification of magma.
Kriging	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Lithological	Description of the physical characteristics of a rock.
Life of mine plans	Plans that are developed for the life of the mine.
Milling	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease	A lease area for which mineral rights are held.
Operating expenditure	Operating expenditures/costs are costs required to operate the mine on a regular basis and includes mine operating costs, process plant operating costs, and general and administrative (G&A) costs
Oxide	Mineral that has undergone chemical reaction in which the substance has combine with oxygen.
Project	A collaborative enterprise, involving research or design, that is carefully planned to achieve a particular aim i.e.

Term	Definition
Sedimentary	Pertaining to rocks formed by the lithification of accumulated of sediments, formed by the erosion of other rocks.
Stratigraphy	The study of stratified rocks in terms of time and space.
Strike	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide	A sulfur bearing mineral.
Sustaining Capital	Capital estimates of a routine nature, which is necessary for sustaining operations.
Tailings	Finely ground waste rock from which valuable minerals or metals have already been extracted.
Thickening	The process of concentrating solid particles in suspension.
Total Expenditure	All expenditures including those of an operating and capital nature.
Variogram	A statistical representation of the characteristics (usually grade).

2.3.4 Abbreviations

Table 2.3, Table 2.4, and Table 2.5 provide lists of abbreviations that are used in this Report.

Table 2.3: Abbreviations

Abbreviation	Unit or Term
A	amperes
AA	atomic absorption
AAS	atomic absorption spectroscopy
ABA	acid base accounting
ACI	American Concrete Institute
ADR	adsorption-desorption-recovery
AIC	American Institute of Constructors
AISC	American Institute of Steel Construction
Ag	silver
amsl	above mean sea level
ANFO	ammonium nitrate-fuel oil
AP	acid potential
~	approximately
ARD	acid rock drainage
As	arsenic
AT	after tax
$ATNPV_{5\%}$	after-tax net present value at a 5% discount rate
Au	gold
AuCN	assays that determine the cyanide soluble gold content
AuFA	assays that determine the total gold content using the fire assay technique
BDR	baseline Data Report
BIOX	biological oxidation of sulfides using bacteria in reactor tanks
BMP	best management practices established by the State of Idaho
°C	degrees Celsius

Abbreviation	Unit or Term
CAPEX	capital expenditures
CCD	counter-current decantation
cfm	cubic feet per minute
CIL	carbon in leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	carbon-in-pulp
CN	cyanide
CO_3	carbonate
COC	chain of custody
CoG	cut-off grade
CSAMT	controlled source audio magneto-tellurics geophysical survey method
°	degree (degrees)
dia.	diameter
EFMC	East Fork of Meadow Creek, commonly known as "Blowout Creek"
EFSFSR	East Fork of the South Fork of the Salmon River
EGL	effective grinding length
EM	electromagnetic geophysical survey technique
EMF	electromagnetic field
EMF	electromotive force
EPCM	engineering, procurement and construction management
EPH	early production high antimony mineralization from Yellow Pine
EPL	early production low antimony mineralization from Yellow Pine
°F	degrees Fahrenheit
FA	fire assay
famsl	feet above mean sea level
Fe	iron (element)
ft	feet
ft^2	square feet
ft^3	cubic feet
ft^3/st	cubic feet per short ton
FOB	free on board
FS	feasibility study, as defined by NI 43-101
g	grams
gal	gallons
g/L	grams per liter
g-mol	gram-mole
gpm	gallons per minute
G&A	general & administration
GCL	geo-synthetic clay liner
GHG	greenhouse gasses
GPS	global positioning system

Abbreviation	Unit or Term
g/st	grams per short ton
g/t, gpt	grams per metric tonne
HCT	humidity cell test
HDPE	high density polyethylene
HERCO	Hermitian Correction model, a statistical analytical tool
HF	Hangar Flats
HFH	Hangar Flats high antimony mineralization
HFL	Hangar Flats low antimony mineralization
HFZ	hidden fault zone at Yellow Pine
Hg	mercury
HMI	human-machine interface
hp	horsepower
HTH	Historic Tailings high grade gold mineralization
HTL	Historic Tailings low grade gold mineralization
HTM	Historic Tailings average grade gold mineralization
HWF	Hanging Wall fault at Yellow Pine
ICP	inductively coupled plasma
ICP AES	inductively coupled plasma atomic emission spectroscopy, an analytical method for assaying
ICP MS	inductively coupled plasma mass spectrometry, an analytical method for assaying
ID	Idaho, where context indicates
ID^2	inverse-distance squared
ID^3	inverse-distance cubed
IMPLAN	Impact analysis for planning
in	Inches
IP	induced polarization geophysical survey technique
IR	infrared
IRR	internal rate of return, a financial measure
kg	kilograms
kg/t	kilograms per metric tonne
koz	thousand troy ounces
kst	thousand short tons
kst/d	thousand short tons per day
kst/y	thousand short tons per year
kV	kilovolts
kW	kilowatts
kWh	kilowatt-hours
kWh/st	kilowatt-hours per short ton
L	liters
lb	pounds
LiDAR	Light Detection And Ranging distance measuring technology
LLDPE	linear low density polyethylene plastic

Abbreviation	Unit or Term
LOM	life-of-mine
m	meters
Ma	million years
MACRS	Modified accelerated cost recovery system
MBR	membrane bioreactor
MCFZ	Meadow Creek fault zone
mg/L	milligrams/liter
mi	miles
mi^2	square miles
MIBC	Methyl isobutyl carbinol
mL	Milliliter or 10^{-3} liters
MLA	mineral liberation analyzer
Mlbs	million pounds
Moz	million troy ounces
Mst	million short tons
Mst/y	million short tons per year
MFZ	Mule fault zone
MW	Megawatts or million watt (where context indicates)
MWMP	Meteoric Water Mobility Procedure (Nevada)
mV	Millivolt or 10^{-3} volts
MVA	megavolt amperes
NAG	net acid generating
NEPA	National Environmental Policy Act of 1969 (as Amended)
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
NNP	net neutralization potential
NP	neutralization potential
NPR	net of process revenue (NPR), defined as NSR less OPEX and G&A
NSR	net smelter return
OHWM	ordinary high water mark
OPEX	operating expenditures
oz	troy ounces
oz/st	troy ounces per short ton
%	percent
P_{80}	80% passing a certain size
PAX	Potassium amyl xanthate
PEA	Preliminary Economic Assessment as defined in NI43-101
PFS	Preliminary Feasibility Study as defined in NI43-101
PLC	programmable logic controller
PMF	probable maximum flood
PoO	Plan of Operations

Abbreviation	Unit or Term
POX	pressure oxidation
ppb	parts per billion
ppm	parts per million
Psi	pounds per square inch
$PTNPV_{5\%}$	pre-tax net present value at a 5% discount rate
QA/QC	quality assurance/quality control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning electron microscopy
QP	NI 43-101 Qualified Person
RCA	riparian conservation area
RC	reverse circulation drilling
RMS CV	root mean squared coefficient of variation, a statistical tool
ROM	run-of-mine
RQD	rock quality designation
SAG mill	semi-autogenous grinding mill
SEC	U.S. Securities & Exchange Commission
sec	seconds
Sb	antimony
SMC	Sag Mill Comminution
SVFZ	Scout Valley fault zone
SG	specific gravity
SIMS	secondary ion mass spectrometry
SODA	spent ore disposal area
SOG	sale-of-gas
SRCE	standardized reclamation cost estimator
st	short tons (2,000 pounds)
st/h	short tons per hour
st/d	short tons per day
st/y	short tons per year
SPLP	synthetic precipitation leachate procedure
TC-RC	treatment charges – refining charges, which are smelter charges
TDS	total dissolved solids
TIC	total inorganic carbon
Ton	short ton of 2,000 lbs
Tonne	metric tonne of 1,000 kg
TSF	tailings storage facility
TSS	total suspended solids
µ	microns, micrometers(one millionth of a meter)
UTM NAD83	Universal Transverse Mercator North American Datum of 1983 geodetic network
UV	ultra-violet light
V	volts
VFD	variable frequency drive

Abbreviation	Unit or Term
VHF	very high frequency
VLF-EM	very low frequency electro-magnetic geophysical survey
W	watts, where context indicates
W	tungsten, where context indicates
WAD cyanide	weak acid dissociable cyanide
WE	West End
WEFZ	West End fault zone
WEO	Est End oxide mineralization
WES	West End sulfide mineralization
WRSF	Waste rock storage facility
XRD	x-ray diffraction
XRF	x-ray fluorescence
Y	year
yd	yards
yd^2	square yards
yd^3	cubic yards
YP	Yellow Pine
YPH	Yellow Pine high antimony mineralization
YPL	Yellow Pine low antimony mineralization

Table 2.4: Agency and Related Legal & Regulatory Abbreviations

Abbreviation	Agency Name & Related Act or Regulation or Term
ASTM	ASTM International, known until 2001 as the American Society for Testing and Materials
BEHS	Bureau of Environmental Health and Safety, Division of Health, Idaho Department of Health & Welfare
BFPP	bona fide prospective purchaser under CERCLA
BLM	Bureau of Land Management, U.S. Dept. of Interior
CERCLA	U.S. Comprehensive Environmental Response, Compensation, and Liability Act (1980, as amended)
CERCLIS	Comprehensive Environmental Response, Compensation, and Liability Information System
CFR	Code of Federal Regulations (US)
CIM	Canadian Institute of Mining, Metallurgy & Petroleum
CIM Standards	CIM definition standards for Mineral Resources and Mineral Reserves prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014
CPO	contiguous property owner under CERCLA
DMEA	Defense Minerals Exploration Administration, Defense Minerals Administration, U.S. Dept. of Interior
DoD	U.S. Department of Defense
EA	Environmental Assessment
EHSP	Environmental Health and Safety Plan
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
EPA	U.S. Environmental Protection Agency
ESA	Environmental Site Assessments under ASTM

Abbreviation	Agency Name & Related Act or Regulation or Term
FAA	U.S. Federal Aviation Administration, U.S. Dept. of Transportation
FCC	U.S. Federal Communications Commission
FLPMA	Federal Land Policy Management Act (1976, as amended)
HAZWOPER	Hazardous Waste Operations and Emergency Response
IDEQ	Idaho Department of Environmental Quality
IDL	Idaho Department of Lands
IDWR	Idaho Department of Water Resources
ID Team	USFS Interdisciplinary Team
IJRP	Idaho Joint Review Process
IRS	Internal Revenue Service
MOU	Memorandum of Understanding under IJRP
MSGP	Multi Sector General Permit
MSHA	Mine Safety and Health Administration, U.S. Dept. of Labor
NEPA	U.S. National Environmental Policy Act (1969, as amended)
NMFS	National Marine Fisheries Service, a division of the National Oceanic and Atmospheric Administration, U.S. Dept. of Commerce
NOAA	US National Oceanic and Atmospheric Administration, U.S. Dept. of Commerce
NPDES	National Pollutant Discharge Elimination System under the Clean Water Act (1972, as amended)
NPL	National Priorities List under CERCLA
OME	Office of Mineral Exploration, USGS, U.S. Dept. of Interior
RCRA	U.S. Resource Conservation and Recovery Act (1976, as amended)
REC	Recognized environmental condition under CERCLA
ROD	Record of Decision
SEC	U.S. Securities & Exchange Commission
SEDAR	System for Electronic Document Analysis and Retrieval
SOP	standard operating procedures designed by the State of Idaho
SPCC	Spill Prevention, Control and Countermeasures Plan
SRB	China's State Reserve Bureau
SWPPP	storm water pollution prevention plan
TESCP	threatened, endangered, sensitive, candidate, and proposed species
TMDL	total maximum daily loads
USACE	U.S. Army Core of Engineers, U.S. Dept. of Defense
USBM	U.S. Bureau of Mines, U.S. Dept. of Interior
USFS	U.S. Forest Service, U.S. Dept. of Interior
USFWS	U.S. Fish and Wildlife Service, U.S. Dept. of Interior
USGS	U.S. Geological Survey, U.S. Dept. of Interior

Table 2.5: Corporate Abbreviations

Abbreviation	Company Name
AAS	American Analytical Services, an assay laboratory
AGP	AGP Mining Consultants Inc.
ALS	ALS Chemex Labs, Ltd., an assay laboratory
Barrick	Barrick Gold Corporation (formerly American Barrick Resources)
BCM	Blue Coast Metallurgy Ltd.
Biomin	Biomin South Africa (Pty) Ltd, a biological oxidation metallurgical laboratory
Bradley	Bradley Mining Co.
BVRR	Boise Valley Railroad
CSA	Canadian Securities Administrators
Dakota	Dakota Mining Company
Dynatec	Dynatec Metallurgical Technologies, a pressure metallurgy laboratory
El Paso	El Paso Mining and Milling
Franco Nevada	Franco Nevada Corporation
Gold Crest	Gold Crest Mines Inc.
HDR	HDR, Inc.
Hecla	Hecla Mining Company
Homestake	Homestake Mining Company
IGHC	Idaho Gold Holding Company, a subsidiary of MGC
IGR	Idaho Gold Resources, LLC, a subsidiary of IGHC
IGS	Idaho Geologic Survey
IMC	Independent Mining Consultants, Inc.
INPR	Idaho Northern Pacific Railroad
IPCo	Idaho Power Company
MCSM	Meadow Creek Silver Mines Company
MGC	Midas Gold Corp.
MGI	Midas Gold, Inc., a subsidiary of MGC
MGIAC	MGI Acquisition Corporation, a subsidiary of MGI
Midas Gold	Unless otherwise specified, one or more of the subsidiaries of MGC
MinVen	MinVen Corporation
MSE	Millennium Science & Engineering, Inc.
MWH	MWH Americas, Inc.
PAH	Pincock, Allen and Holt
Pegasus	Pegasus Gold Corporation
Pioneer	Pioneer Metals Corporation
Ranchers	Rancher's Exploration Company
SGS	SGS Minerals Inc.
SMI	Stibnite Mines Inc., a subsidiary of MinVen and later Dakota
SRK	SRK Consulting (Canada), Inc.
Strata	Strata, a professional services corporation
Superior	Canadian Superior Mining (U.S.) Ltd.

Abbreviation	Company Name
URS	URS Corporation
Vista	Vista Gold Corp.
Vista US	Vista Gold US Inc., a subsidiary of Vista

Table 2.6: Standard Core Hole Diameters

Table 2.6 presents standard core hole and core size dimensions referred to in this Report. The conversions have been rounded to the nearest approximate whole fraction of an inch.

Size	Hole (outside) diameter	Core (inside) diameter
EX	37.7mm (1-1/2 in)	21.4mm (7/8 in)
AQ	48mm (1-7/8 in)	27mm (1-1/16 in)
AX	48mm (1-7/8 in)	30 mm (1-3/16 in)
BQ	60mm (2-3/8 in)	36.5mm (1-7/16 in)
BX	60mm (2-3/8 in)	42.1mm (1-5/8 in)
NQ	75.7mm (3 in)	47.6mm (1-7/8 in)
NX	75.7mm (3 in)	54.8mm (2-5/32 in)
HQ	96mm (3-3/4 in)	63.5mm (2-1/2 in)
PQ	122.6mm (4-13/16 in)	85mm (3-3/8 in)

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3 RELIANCE ON OTHER EXPERTS

The Stibnite Gold Project Report relies on reports and statements from legal and technical experts who are not Qualified Persons as defined by NI 43-101. The Qualified Persons responsible for preparation of this Report have reviewed the information and conclusions provided and determined that they conform to industry standards, are professionally sound, and are acceptable for use in this Report.

3.1 PROPERTY OWNERSHIP AND TITLE

Legal review of the Stibnite Gold Project property ownership and title was completed by the Idaho law firm Givens Pursley LLP (Givens Pursley). Givens Pursley commissioned multiple Landman Reports for the Project that cover various patented and unpatented mining claims. The Landman Reports were completed in accordance with reasonable industry standards by person(s) having appropriate training, experience and expertise. Givens Pursley's review of these Landman Reports and related examinations are summarized in various title opinion documents, the most recent of which were completed in 2013. Givens Pursley concluded that Midas Gold is vested with fee simple, mineral, or possessory record title to, or an option to purchase, the Stibnite Gold Project properties described in Section 4 of this Report, subject to the royalties, agreements, limitations and encumbrances described in Section 4.4.

3.2 AUTOCLAVE DESIGN AND SIZING

Technical assistance for the design and sizing of the autoclave was provided by Mr. Herman Pieterse, engineering consultant, who has 25 years of experience in autoclave operation and design. Mr. Pieterse provided significant input concerning the design, operation, and sizing of the autoclave and appurtenant equipment for pressure oxidation of sulfide concentrates for the liberation of gold. The pressure oxidation process is critical to the recovery of gold from the Stibnite Gold mineralization.

3.3 WATER RIGHTS

Mr. Terry Scanlan, P.E., P.G. of SPF Water Engineering, LLC performed a comprehensive review of Midas Gold's water rights portfolio. The water rights held by Midas Gold are summarized in Section 5.5.2 of this Report.

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4 PROPERTY DESCRIPTION AND LOCATION

4.1 MINERAL TITLE

Midas Gold's property holdings consist of patented lode claims, patented mill site claims, unpatented federal lode claims and unpatented federal mill site claims (collectively, "Claims") which cover approximately 27,104 acres (approximately 42 mi^2) as shown on Figure 4.1. Appendix II presents a detailed mineral concession summary, a land status map, and complete tables listing the Claims. No significant flaws or title issues have been identified in multiple formal title reviews of the Claims performed by a qualified, independent, title examiner. A number of independent legal opinions in respect of mineral title have been prepared on behalf of Midas Gold in support of its initial listing as a public company, subsequent financings, and sale of a royalty to a third party.

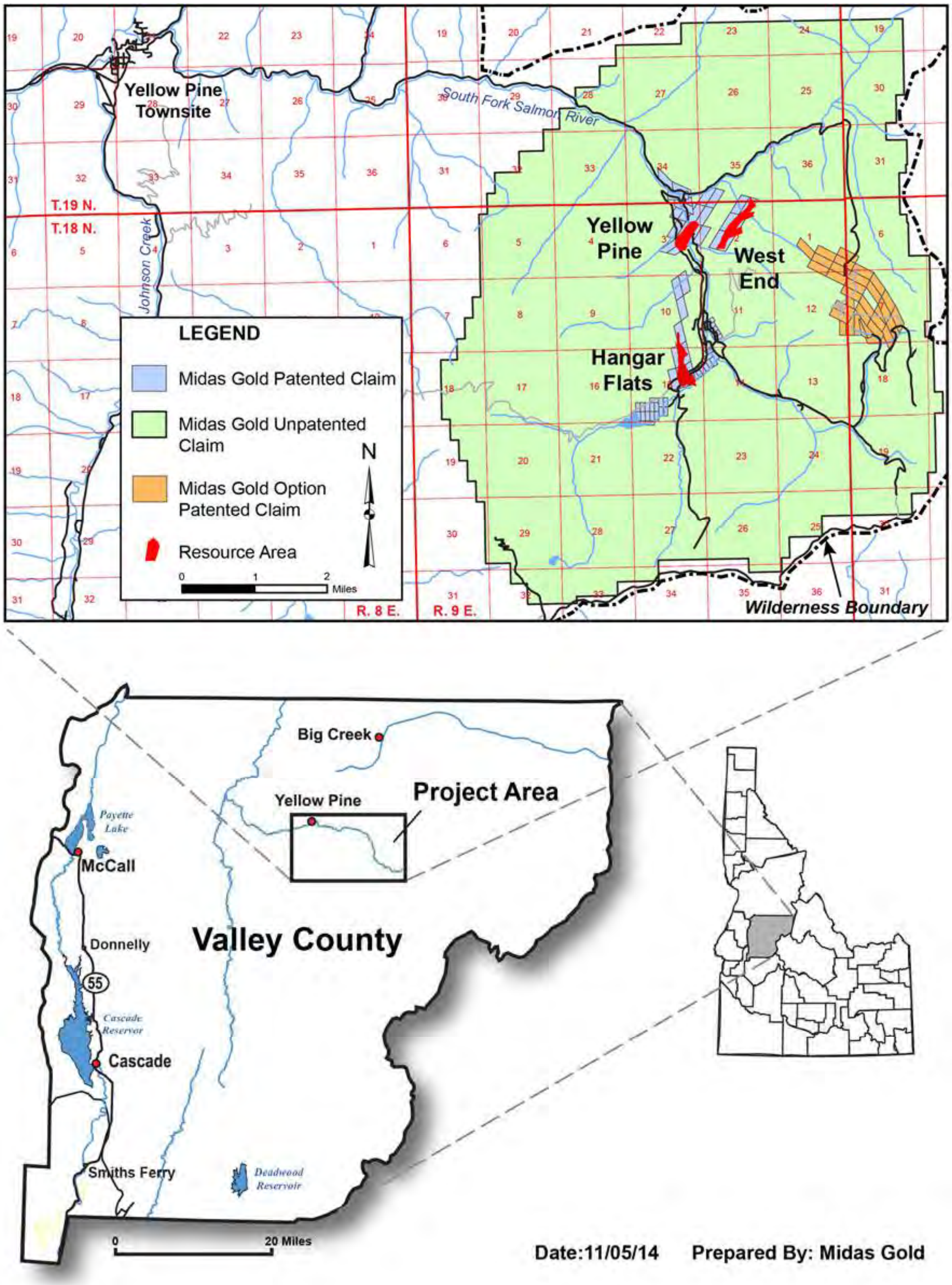
4.2 LOCATION

The Project is located in central Idaho approximately 98 mi northeast of Boise, Idaho, 40 mi east of McCall, Idaho, and approximately 10 mi east of Yellow Pine, Idaho (Figure 4.1) in all or part of the following sections (Boise Meridian):

- Township 19 North, Range 9 East, Sections 21 to 28 and 32 to 36;
- Township 19 North, Range 10 East, Sections 19, 30, and 31;
- Township 18 North, Range 9 East, Sections 1 to 30 and 32 to 36; and
- Township 18 North, Range 10 East, Sections 5 to 8, 17 to 20, 29 and 30.

The Project area elevations range from approximately 6,500 ft to over 8,900 ft above sea level and is centered at latitude 44°54'25" N and longitude 115°19'37" W and, in State Plane Idaho West coordinates, at 1103 1181270 ft US N and 1103 2734259 ft US W.

Figure 4.1: Project Location Map



4.3 NATURE AND EXTENT OF TENURE

The following description was updated to July 30, 2014. Claim groups under full Midas Gold ownership are discussed in this section while those with encumbrances are detailed in section 4.4.

Midas Gold acquired 229 federal unpatented claims by purchase from previous owners in 2009 and 2011. These include 46 federal mill site claims which carry surface use rights but no mineral rights, and 183 federal unpatented lode-mining claims. In addition to the purchased claims, Midas Gold acquired, by staking on its own behalf, an additional 238 federal unpatented lode mining claims in 2009, 921 federal unpatented lode-mining claims in 2011, and one federal unpatented lode-mining claim in 2012 (re-staked to correct a BLM clerical error). A complete list of active claims is included in Appendix II. Federal unpatented claims total approximately 25,762 acres. Maintenance of unpatented federal claims requires that Midas Gold provide a list of claims and serial numbers to the Bureau of Land Management (**BLM**) with annual maintenance fees of $155 for each lode or mill site claim on or before September 1st each year. This was completed for the most recent filing year on August 4, 2014, and an Affidavit of Satisfaction was subsequently recorded in Valley County on August 21, 2014. There is no underlying royalty on these federal lode and mill site claims other than the Franco-Nevada Corporation (**Franco-Nevada**) royalty detailed in Section 4.4. None of the Claims are subject to back-in rights.

The ownership of the Yellow Pine Deposit was conveyed to Midas Gold in 2011 by way of a company merger between a subsidiary of Midas Gold Corp. and a subsidiary of Vista Gold Corp. (**Vista**) agreed to February 22, 2011. As a result of the combination, Midas Gold, Inc. (**MGI**) became a wholly owned subsidiary of Midas Gold Corp. The Yellow Pine claim group includes 17 patented lode mining claims totaling approximately 301 acres and eight unpatented lode-mining claims (already included in the unpatented total above).

On April 28, 2011, Midas Gold purchased 6 patented lode claims in the eastern area of the Project. This group of claims is referred to as the Fern claim group, totaling approximately 100 acres.

4.4 ROYALTIES, OPTION AGREEMENTS AND ENCUMBRANCES

4.4.1 Royalties and Option Agreements

On June 11, 2009, Midas Gold acquired an option to purchase the Meadow Creek group of patented lands from Bradley Mining Co. (**Bradley**) by direct purchase of nine patented mining claims, totaling approximately 184 acres. These lands are subject to a 5% net smelter return (**NSR**) royalty interest (the **Oberbillig Royalty**) to the Oberbillig Group, a group of beneficiaries to the royalty. However, on May 27, 2009, Midas Gold entered into the Oberbillig Royalty agreement whereby it purchased the full 5% NSR royalty from the Oberbillig Group. Midas Gold has an underlying promissory note in respect to the agreement in favor of the Oberbillig Group, which has not reached maturity or been settled as of the date of this Report and is secured by the Oberbillig Royalty. The principal balance on the promissory note is $160,000 as of the date of this Report. The promissory note accrues interest at 3% per annum and matures on June 2, 2015. Property taxes for this and other patented claim groups are included in Appendix II.

Midas Gold secured a purchase agreement from the J.J. Oberbillig Estate on June 2, 2009 to acquire 30 patented federal mill site claims totaling approximately 149 acres, which include both surface and mineral rights and six patented federal lode claims totaling approximately 124 acres. The majority of the mineralization constituting the West End Deposit is located within these 6 patented lode claims. The surface right for portions of six of the patented federal mill site claims was granted to Hecla Mining Company (**Hecla**), however the mineral rights, and the right to explore and mine were retained by the J.J. Oberbillig Estate. With respect to the purchase option agreement described above, Midas Gold currently owes a final annual payment of $40,000 to the J.J. Oberbillig Estate, due June 2, 2015.

On May 3, 2011, Midas Gold entered into an option to purchase 27 patented lode claims totaling approximately 485 acres from the J.J. Oberbillig Estate (the Cinnabar option claims). The total purchase price of these claims is $750,000. To date, Midas Gold has paid Oberbillig a total of $450,000 ($150,000 at closing and $100,000 for the first three of five one-year extensions). All monies spent to date apply toward the purchase price should Midas Gold decide to exercise the option. Two more one-year extensions are available with the final extension expiring on May 1, 2017. Midas Gold is currently responsible for property taxes on these claims. Property tax information for all claim groups is included in Appendix II.

Effective May 9, 2013, Midas Gold granted a 1.7% NSR royalty on future gold production to Franco-Nevada, but not antimony and silver. The royalty agreement applies to all patented and unpatented mineral claims, with the exception of the Cinnabar claim group where Midas Gold holds an option to purchase at this time. Midas Gold also retains an option to re-acquire one-third of the royalty for $9.0 million (this option expires May 9, 2016).

4.4.2 Consent Decrees under CERCLA

Several of the patented lode and mill site claims acquired by Midas Gold comprising part of the West End Deposit, and the Cinnabar claims held under option from the Estate of J.J. Oberbillig are subject to a consent decree entered in the United States District Court for the District of Idaho (United States v. Estate of J.J. Oberbillig, No. CV 02-451-S-LMB (D. Idaho)) in 2003, involving or pertaining to environmental liability and remediation responsibilities with respect to the affected properties described therein. This consent decree provides the regulatory agencies that were party to the agreement access and the right to conduct remediation activities under their respective Comprehensive Environmental Response, Compensation, and Liability Act (CERCLA) and Resource Conservation and Recovery Act (RCRA) authorities as necessary and required to prevent the release or potential release of hazardous substances. In addition, the consent decree requires that heirs, successors and assignees refrain from activities that would interfere with or adversely affect the integrity of any remedial measures implemented by government agencies.

The mineral properties held by Midas Gold's subsidiary, Idaho Gold Resources and that portion of Midas Gold's mineral properties acquired from Bradley pursuant to the Bradley Mining Agreement (i.e. collectively, the Hangar Flats Deposit and Yellow Pine Deposit) are subject to a consent decree that was entered in two United States District Court cases (United States v. Bradley Mining Co., No. 3:08-CV-03986 TEH (N.D. Cal.) and United States v. Bradley Mining Co., No. 3:08-CV-05501 TEH (N.D. Cal)). The first case concerned Bradley's Sulfur Bank Mercury Mine Superfund Site in Lake County, California while the second case was related to the Stibnite Mine site in Valley County, Idaho (part of the Project). On December 7, 2011, these two cases were consolidated into one case (United States v. Bradley Mining Co., No. 3:08-CV-03986 TEH (N.D. Cal.)). A proposed consent decree was lodged on February 14, 2012 and approved on April 19, 2012 after appropriate public comment. The consent decree includes a financial order against Bradley and related terms. The consent decree also states that if Environmental Protection Agency (EPA) or the Forest Service determines that "land/water use restrictions in the form of state or local laws, regulations, ordinances or other governmental controls are needed to implement response activities at the Stibnite Mine Site, ensure the integrity and protectiveness thereof, or ensure non-interference therewith" Bradley Mining agrees to cooperate with EPA's or the Forest Service's efforts to secure such governmental controls.

The Corporation cannot ensure it has identified every consent decree or administrative order which may affect the Stibnite Gold Project.

A "bona fide prospective purchaser" defense is a legal defense available to an owner who, after conducting appropriate inquires, establishes that environmental liability occurred before the owner acquired the property. Midas Gold has taken and will continue to take all steps required to establish itself as a bona fide prospective purchaser.

4.5 ENVIRONMENTAL LIABILITIES

A number of environmental studies and regulatory investigations in the District have identified numerous areas of potential environmental degradation related to historic mining. For detailed ownership and mine development history in the District refer to Section 6 of this Report. In 2009 and 2010, Midas Gold and Vista US contracted Millennium Science & Engineering, Inc. (MSE) to conduct Phase I and Phase II Environmental Site Assessments (ESAs), as prescribed by the American Society for Testing and Materials (ASTM) Standard Practices for Environmental Site Assessments. The results of the ESAs indicate that overall water quality in all drainages is good given the duration and extent of mining. MSE's Phase I ESA identified 88 potential or known Recognized Environmental Concerns (RECs) which included several redundant items (e.g. RECs that span both patented and unpatented property boundaries are counted more than once). Based strictly on location or legacy site features, there are approximately 24 distinct RECs that Midas Gold continues to evaluate on an individual basis. There are also some non-ASTM (e.g., geotechnical) issues that are counted in MSE's REC total. The following sections describe the existing historical liabilities; Figure 4.2 provides the general location and extents of these liabilities.

4.5.1 Spent Ore Disposal Area and Historic Tailings

Bradley placed an estimated 4,000,000 cubic yards of mill tailings in the upper Meadow Creek Valley from 1946 to 1952. At the time, Meadow Creek was diverted around the tailings, however, in 1959, Bradley was ordered to breach the diversion and allow the creek to resume a more natural course through the tailings. Over the next 20 years an estimated 10,000 cubic yards of tailings were eroded and carried downstream.

In the 1980s, Canadian Superior Mining (Superior) was required to mitigate these historic tailings by constructing a new Meadow Creek diversion channel and stabilizing the tailings by covering them with neutralized ore from their on-off heap leach operations, creating the Spent Ore Disposal Area (SODA). Superior and their successors placed an estimated 6,050,000 tons of spent ore here between 1982 and 1994. The Meadow Creek diversion was moved again circa 2000.

The majority of the historic tailings are located below the water table and likely continue to leach metals. In addition, the upstream wetland lying west of the historic tailings and SODA (formerly a water storage area related to Bradley's operations) is also underlain by tailings.

4.5.2 Hangar Flats / Former Meadow Creek Mine

The first claims were staked in this area in 1914, but significant development of the underground Meadow Creek Mine began c. 1927 and the adjacent Bradley (née, Yellow Pine Company) mill beginning production in 1932. The underground mine closed in 1938 but the mill continued to operate processing ore from the Yellow Pine Pit located 2.5 miles to the north. In 1945, the crusher was moved from the mill to an in-pit location at Yellow Pine and, in 1949, a smelter was added adjacent to the mill at Meadow Creek. The mill closed in 1952 and, by 1957, was dismantled or abandoned. Up until 1946, mill tailings were placed in impoundments adjacent to the mill or pumped directly into Meadow Creek during the winter months. The US Forest Service (USFS) has performed several remedial actions in this area, including removal of some historic tailings and the smelter stack in 2003, re-channelization of lower Meadow Creek in 2005, and covering the 2003 impoundments with clean fill in 2009. In spite of these actions, historic tailings remain buried over much of the area, including under the airstrip and adjacent to Meadow Creek.

Later heap leach gold operations also operated in the area of the former Meadow Creek Mine, mill and related facilities, in what is now the area of the current Hangar Flats deposit. Superior and their successors operated on-off heap leach pads and an adjacent process plant from 1982-1997, processing ore from the West End area. The empty leach pads still exist but have been covered by fill. The former process plant and related facilities (site of the current Midas Gold helicopter hanger) have been removed, however, subsurface impacts remain including a former diesel fuel release. Hecla also had a gold heap operation here from 1988-1992 processing ore from the Homestake area of the Yellow Pine deposit. The loaded heap, underdrain system, and infiltration galleries remain but Hecla's former processing facilities have been removed.

4.5.3 Garnet Pit

The Garnet Pit was a short-lived open pit mine, operated by Stibnite Mines Inc. (SMI) in the mid-1990s, and located just east of Midas Gold's current camp and core shed area. The ore was processed on the Superior leach pads. The open pit and associated waste rock storage facilities cover an area of approximately 5.5 acres.

4.5.4 Former Stibnite Town Site

During World War II, Stibnite was an incorporated town with a peak population of over 750 Bradley employees and their families. The town site area shown on Figure 4.2 included several employee homes, the recreation center, the hospital, the school, the automobile service station, and the municipal waste landfill. Other "neighborhoods" existed in Stibnite outside this area, including Fiddle Creek and Midnight Creek and there was an earlier landfill location approximately ½ mile to the east.

4.5.5 Former EFSF Haul Road and Adjacent Areas

The area from the former Stibnite town site to the Yellow Pine Pit is the site of many legacy environmental features. The Monday Camp, adjacent to the Yellow Pine Pit, was the site of Bradley's truck repair, machine, and maintenance shops. Fuel oil storage was significant and petroleum contamination is likely to be encountered in the vicinity. Other areas along the now-reclaimed haul road were the site of the former Bradley saw mill, various man camps, the former SMI pilot plant, and other activity. Midas Gold's current exploration camp / core shed area was once the site of the SMI crusher, staging area, shop, and fuel depot. Hecla also maintained a camp and an equipment staging area here.

4.5.6 Yellow Pine Pit and Homestake

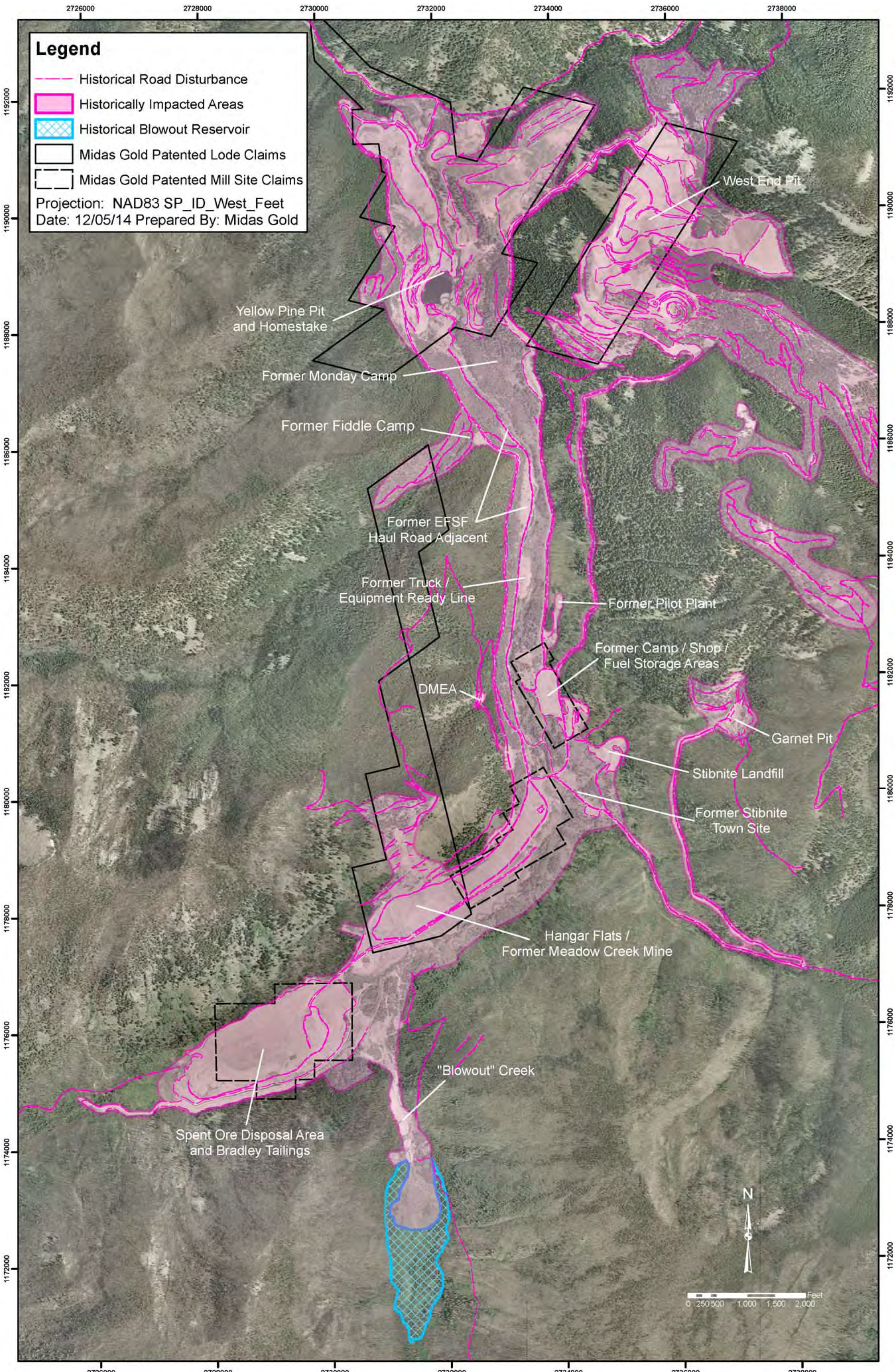
Open pit mining in the Yellow Pine Pit began in 1937. During World War II and the Korean War, the pit provided an estimated 90% of the antimony and 50% of the tungsten needed for U.S. war efforts. In 1943, the 3,500 foot Bailey Tunnel was driven to divert the East Fork of the South Fork of the Salmon River (EFSFSR) away from the open pit. Mining ceased in 1952 until Hecla returned to mine the adjacent Homestake area from 1988-1992.

The open pit and subsequently the pit lake has been a barrier to fish passage since 1938. The pit lake has also acted as a sediment trap and holds legacy tailings eroded from the Meadow Creek Valley upstream. There are numerous waste rock storage facilities throughout this area and spent ore has been placed here and elsewhere in the District (e.g., as road surfacing). A waste repository was also created on the west rim of the pit in 2003 during a Forest Service removal action related to the former smelter stack and other remediation actions in the Hangar Flats area.

4.5.7 West End

West End is a series of open pits and waste rock storage facilities. Superior began work in the area in 1982. They and their successors (Pioneer Metals Corp., Pegasus Gold Corp, Dakota Mining, and SMI) continued mining until 1997. The area remains sparsely vegetated. The upper reach of West End Creek is in a large diameter culvert pipe beneath a waste rock storage facility.

Figure 4.2: Legacy Environmental Liabilities



4.5.8 East Fork of Meadow Creek

A small reservoir was constructed on the East Fork of Meadow Creek (commonly referred to as Blowout Creek) in 1931 for electric power generation. The dam was raised in 1949 to accommodate increased water demand created by the new smelter. In June 1965, high runoff caused a catastrophic failure of the dam. Damage at the time was significant and the drainage remains a major source of sedimentation due to active head cutting and erosion during high flow events. The head cutting has also lowered the water table in the upper valley, reducing the quality of the wetlands in that area.

4.5.9 DMEA

As the first generation of active mining ceased in 1952, Bradley was awarded two contracts by the Defense Minerals Exploration Administration (DMEA) and performed exploration work through 1955. This work included 4,900 ft of underground workings with an associated adit and waste rock storage facility at the location shown on Figure 4.2. A creek runs through the waste rock storage facility.

4.5.10 Underground Workings

In addition to the disturbed areas show on Figure 4.2, several underground workings also exist. There are numerous tunnels, adits, shafts, and associated waste rock storage facilities. Major historic underground workings include the Meadow Creek Mine, the Cinnabar Tunnel, the Monday Tunnel, the North Tunnel, the Bailey Tunnel (former EFSFSR water diversion), and DMEA workings; while on the adjacent optioned claims, the Cinnabar Mine has extensive underground workings and related surface disturbance.

4.5.11 Past Remediation

In the past, regulatory actions under U.S. Comprehensive Environmental Response, Compensation, and Liability Act (CERCLA), U.S. Resource Conservation and Recovery Act (RCRA), and state law have been taken by the EPA, USFS, the Idaho Department of Lands (IDL) and the Idaho Department of Environmental Quality (IDEQ) against historic mining operators. These agencies along with several past owners and operators have conducted certain remediation of historic mining activities at the Project site. The primary projects include the following:

- stabilizing and covering with spent ore the 551 acres historic tailings pile at the upper-south end of the Meadow Creek valley;
- re-routing the Meadow Creek diversion and reconstructing it to accommodate a 500-year flood event;
- capping some of the old heap leach facilities;
- removal of nearly all above ground historic mining structures and facilities including the former Meadow Creek mill and smelter;
- removal of leaking underground fuel storage tanks left from previous owners; and
- EPA conducted and paid for the clean-up of the smelter stack, assay labs and other sites at Stibnite well after 2001.

The Stibnite site was considered and rejected for listing on the National Priorities List (NPL) in September 2001.

Midas Gold may conduct environmental enhancement and reclamation opportunities in the future as part of an operating plan, including potentially improving or re-establishing key fish passage routes.

4.6 PERMIT REQUIREMENTS

4.6.1 Exploration Permits and Status

The Midas Gold exploration programs completed to date consisted of road and drill pad construction to support drilling on both public and private lands. There are different permitting requirements for activities on the respective public and private land holdings.

The USFS, Payette National Forest, Krassel Ranger District has jurisdictional authority over mitigating surface disturbance associated with exploration and mining-related activities on public lands within its administrative area. Although some of the claims are in the Boise National Forest, the Payette National Forest has been granted administrative authority for the entire Project area. IDL, Payette Lakes Area District has jurisdictional authority over exploration and mining-related activities on private lands (as well as oversight on activity on public lands as well) within its administrative area.

Midas Gold is currently conducting exploration in the Stibnite mining district on patented property under an annual IDL Notice of Exploration. Midas Gold currently has an exploration Plan of Operations (PoO) filed with the USFS under POO-2014-049059 and is awaiting approval.

4.6.2 Exploration Compliance Evaluation

Midas Gold is in full compliance with applicable laws and regulations related to its exploration activities. The staff of IDL, USFS, U.S. Fish and Wildlife Service (USFWS), EPA, IDEQ, and National Marine Fisheries Service (NMFS) have toured the Project site several times during ongoing activities and have issued required permits and granted approval for Midas Gold's activities on the site.

4.6.3 Mine Development Permits

The environmental permitting process for the development of a mine within the Project boundaries would primarily involve: water quality permits, wetlands permits, surface and ground water use permits, authorizations to relocate stream channels, permits addressing design and construction of a tailings dam, air-quality permits, a cyanide use permit, and approval of a final PoO/Reclamation Plan. In total, over 30 separate local, state, and federal environmental permits and licenses would be required to construct and operate a mine within the Project boundaries.

A detailed list of applicable permits, licenses, and approvals is listed in Table 4.1. The National Environmental Policy Act (NEPA) requires federal agencies to study and consider the likely environmental effects of any project before allowing it to proceed. In the case of the development of a mine within the Project boundaries, the major federal action necessary to move the project forward is the approval of the final Plan of Operations/Reclamation Plan by the USFS (Forest Supervisor) in conjunction with the IDL and other cooperating agencies. This would be done by the preparation, review, and completion of a decision (Record of Decision or ROD) on the Environmental Impact Statement (EIS). The EIS would likely be prepared by a qualified third-party environmental contractor under the guidance of the USFS. The cost of the EIS would be borne by Midas Gold.

A more detailed description of all the permits and authorizations listed in Table 4.1 is provided in Section 20. Some ancillary permits and licenses, not listed in the table that may be required are conditional use permits, rights of way and Federal Communications Commission (FCC) communications licenses.

Table 4.1: List of Permits, Licenses, and Approvals Required for the Project

Permit, License, or Approval	Purpose of Authorization	Timing
USFS		
Approval Final Plan of Operations (36 CFR 228 Subpart A)	To allow for locatable mineral exploration and development. This PoO must be consistent with the forest plan. Approval follows the EIS ROD and incorporation of specified requirements and mitigation and monitoring in the final plan so as to minimize or eliminate effects on surface resources. A reclamation plan and financial assurance are a primary element of the final PoO. A waste management plan would also be important.	To be filed after the EIS ROD. Requires input from the final feasibility study.
Road Use Permit	To specify operation and maintenance responsibilities on forest roads. Valley County is also involved as a primary agency. The Road Use Permit would be incorporated into the final PoO.	Required annually for current exploration and final Plan of Operations.
Mineral Material Permit	To allow Midas Gold to collect and use borrow materials from national forest lands.	Would be needed during construction.
Timber Sale Contract	To allow Midas Gold to harvest commercial timber from the Project area (construction clearing). The Timber Sale Contract would be incorporated into the final PoO.	This is a pre-construction need.
Cultural Resource Clearance	To obtain joint approval from the USFS and State Historic Preservation Officer prior to construction.	This is a pre-construction authorization.
Monitoring Plans	Part of the final PoO to assure compliance with state and federal environmental standards.	Required for construction, operations, reclamation and post-closure.
Plan of Operations Review	To ensure consistency with design of plant processing, waste management, water treatment, access roads and other facilities, operational requirements as described in the ROD, final PoO, and other permit approvals.	This is an annual requirement.
U.S. Environmental Protection Agency		
NPDES Permit (water discharge) (EPA)	Required under Section 402 of the Clean Water Act for point source discharges to waters of the US, total maximum daily loads must be considered in this permit. Also required for all "new sources". Section 401 Certification of NPDES and Corps 404 Permits by IDEQ is also required.	Application must be filed 180 days prior to discharging.
Storm Water Pollution Prevention Plan (EPA)	Required to minimize or mitigate the effects of storm water discharges; also includes snowmelt runoff and surface runoff and drainage.	This is a pre-construction requirement.
Section 311 Contingency Plan (EPA)	Required to develop a spill prevention plan for above-ground fuel storage capacity in excess of 1,320 gallons (5,000 liters), or below-ground storage greater than 42,000 gallons (159,000 liters).	Needed during construction, operations and post-closure.
Drinking Water Act Underground Injection Control Permit (EPA)	To regulate subsurface emplacement of wastewater by well injection; may also apply to land application of wastewater.	This permit may be required for operations.

Permit, License, or Approval	Purpose of Authorization	Timing
Prevention of Significant Deterioration Determination (EPA)	Initial analysis to demonstrate the mine would not emit more than 250 tons (230 tonnes) of fugitive dust per year. This is a preconstruction authorization.	To be evaluated as part of the EIS.
Permit to Construct/Permit to Operate (Jointly with IDEQ)	Required to be in compliance with applicable requirements of the Clean Air Act to the extent the permit specifically includes these requirements, or includes a determination that specific requirements do not apply.	To be filed during the EIS/pre-construction phase.
USACE		
Section 404 Dredged and Fill Permit	Required for discharge of dredged or fill material into waters of the U.S. including wetlands.	Concurrent with the EIS; ROD required.
NOAA Marine Fisheries / USFWS		
Fish and Wildlife Endangered Species Act Consultation	Required to demonstrate that the proposed action would not likely jeopardize threatened and endangered fish and wildlife species.	Concurrent with the EIS; ROD required.
Mine Safety and Health Administration (MSHA)		
Safety Plan	This plan is developed to assure the health and safety of the nation's miners. The plan addresses accident protection, communications, prohibition of alcohol and drugs, and other considerations.	Required for active mining operations.
Executive Orders (USFS, EPA, USFWS, and USACE)		
E.0.11988; E.0.11990	These two executive orders deal with the protection of floodplains and floodplain management, and the protection of wetlands.	Concurrent with the EIS.
Indian Tribes		
Native American Consultation	Required to ensure no significant impacts on Nez Perce, Shoshone-Bannock, and Shoshone-Paiute traditional cultural values, practices, properties, or human remains.	Concurrent with the EIS.
IDEQ		
Rules Governing Cyanidation Facilities (Jointly with IDL)	Required to construct and operate cyanide processing facilities; rules also address performance bonds for reclamation and permanent closure of these operations.	To be filed after EIS scoping.
Wastewater Land Application Permit	To regulate the application of industrial/municipal wastewater to plan for the purpose of treatment; used to meet zero discharge requirements.	To be filed after EIS scoping.
Ground Water Rule	Required to protect local ground water resources and quality to be maintained at or near background levels (sites-specific standards).	To be filed after PDEIS.
Air Quality Tier Permit	Required to operating permit for major stationary sources of air pollutants.	To be filed concurrent with EIS.

Permit, License, or Approval	Purpose of Authorization	Timing
Idaho Department of Water Resources (IDWR)		
Water Rights	These permits are required prior to the diversion and use of surface or ground.	Additional to be filed after PFS.
Stream Channel Alteration Permit	Required to divert a stream around mining facilities (i.e. EFSFSR around Yellow Pine open pit)	To be filed prior to PDEIS.
Dam Safety Permit	Required for the construction of a tailings dam; includes detailed design and specifications.	To be filed prior to PDEIS.
IDL		
Reclamation Plan	Describes plans for closing and reclaiming components of the operation located on patented (fee) ground. This plan and associated financial assurance closely interface with the final PoO for the USFS.	To be filed prior to PDEIS.
State of Idaho – Historic Preservation Office		
State Historic Preservation Officer Consultation	The final PoO must receive clearance by the State Historic Preservation Officer.	Approval required prior to construction.
Valley County Planning Department		
Consistent with Comprehensive Plan	The final PoO (proposed Project) must be consistent with the goals and objectives of the Valley County Comprehensive Plan.	Approval required prior to construction.
Valley County Building Department		
Individual Building Permits	A number of building permits would need to be obtained for the primary Project components	Approval required Prior to construction.
Valley County Road Department		
Road use Permits	Road use permits would be required by the county annually; a user fee to pay for necessary road maintenance would be negotiated, including special conditions for winter use.	Annual application required.
Other Consultations/Clearances		
Other Consultations	These include, among others: Executive Order 12898 Environmental Justice, Migratory Bird Treaty Act, Bald Eagle and Golden Eagle Protection Act, USFS Mining Regulations (36 CFR 228A).	Part of the EIS process.

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5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 TOPOGRAPHY, ELEVATION AND VEGETATION

The Project is located within the Salmon River Mountains of Central Idaho. The area consists of uplifted rocks of the Idaho Batholith deeply incised by the East Fork of the South Fork of the Salmon River (EFSFSR). The area is comprised of steep, rugged, and forested mountains with narrow, flat valleys at an elevation of approximately 6,500 ft and nearby mountains rise to an elevation of approximately 7,800 to 8,900 ft. The land is heavily wooded with fir and pine trees and underbrush is common. Large forest fires burned much of the area in 2002, 2006 and 2007. Photograph 5.1 and Photograph 5.2 depict local topography, vegetation, and surface features.

Photograph 5.1 View Looking South Along the EFSFSR

Source: SRK, 2012

Photograph 5.2: View Looking West up Meadow Creek

Source: SRK, 2012

5.2 CLIMATE AND LENGTH OF OPERATING SEASON

The climate is characterized by moderately cold winters and mild summers. Most precipitation occurs as snowfall in the winter and rain during the spring. The local climate allows for year-round operations as evidenced by historic production and climate information.

Weather records indicate that the average precipitation (equivalent rainfall) is approximately 32.19 inches per year. Average temperatures and precipitation are shown in Table 5.1.

Table 5.1: Project Climate Data

Month	Average Temperature (˚F)	Average Precipitation (in)
January	20.1	4.1
February	21.8	3.3
March	27.7	3.5
April	32.9	3.0
May	40.7	2.6
June	48.7	2.1
July	58.1	1.0
August	56.5	1.0
September	48.7	1.8
October	39.2	2.1
November	26.3	3.7
December	18.8	4.0
Average	36.6	32.2

5.3 ACCESS TO PROPERTY

The property is located approximately 152 road-miles northeast of Boise, Idaho, a city with a population of more than 600,000 people in its metropolitan area. Figure 5.1 shows a map of current access routes.

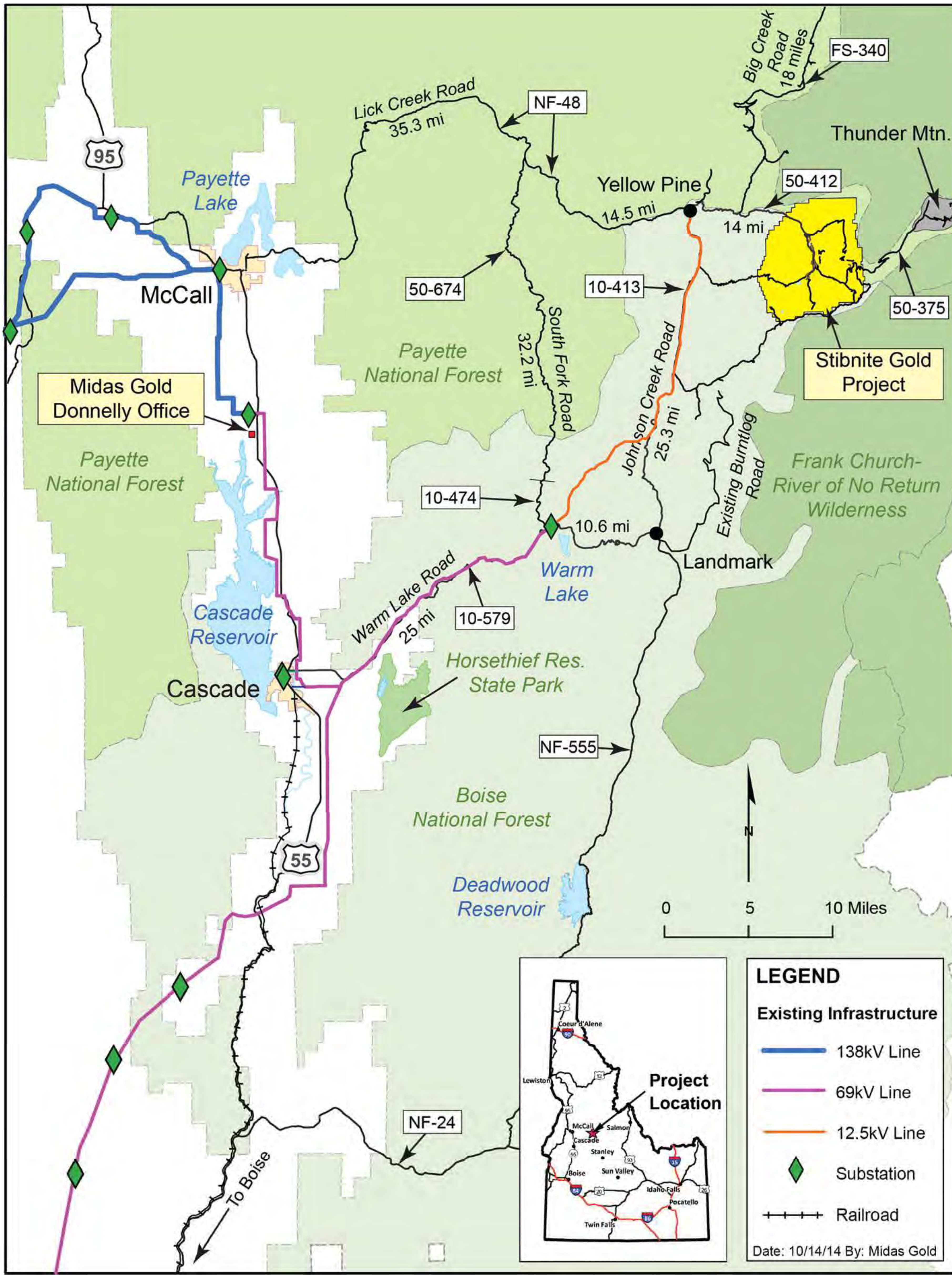
The primary access to the Project area is via:

- Boise to Cascade – Highway 55 (77.4 mi)
- Cascade to Landmark – two-lane, paved Warm Lake Road (35.6 mi)
- Landmark to Yellow Pine – single-lane, unpaved Johnson Creek Road (25.3 mi)
- Yellow Pine to Stibnite – single-lane, unpaved Forest Service 50-412 Road (14 mi)

The primary access (the "Johnson Creek Route") measures approximately 84 mi from Cascade to Stibnite and is not available at certain times of the year when Johnson Creek Road is impassable due to snow. Alternate, low elevation, year-round access is available by traveling from Cascade along the Warm Lake Road and turning north on the South Fork Road 10.6 mi west of Landmark and then turning east onto the East Fork Road (NF-48) towards Yellow Pine and onto Stibnite (the "South Fork Route"). The distance from Cascade to Stibnite is approximately 86 mi along this alternate South Fork Route.

Another route available in snow-free months starts by travelling east on Lick Creek Road near McCall, Idaho, towards Yellow Pine and onto Stibnite (the "Lick Creek Route"). The distance from McCall to Stibnite along the Lick Creek Route is approximately 63 mi. and approximately 93 mi from Cascade to Stibnite via McCall. A grass airstrip is located at Johnson Creek, about 3 mi south of the town of Yellow Pine and a 2,300 ft long improved gravel airstrip is located at Stibnite.

Figure 5.1: Site Access and Existing Pertinent Regional Infrastructure



5.4 SUFFICIENCY OF SURFACE RIGHTS

Midas Gold currently controls 27,105 acres of land through a combination of 1,492 patented and unpatented claims. Surface facilities associated with development of the Stibnite Gold Project would be located on a combination of public and private property under rights established by the 1872 Mining Law, current USFS regulation, and IDL regulations for private property mining development. Approvals for such development come from approval of the USFS Plan of Operations (PoO), which will come in the form of a Record of Decision following completion of an anticipated Environmental Impact Statement, along with a mined land reclamation plan from the IDL. Additional information on Midas Gold patented and unpatented claims are provided in Section 4; additional information on permitting is included in Section 20.

5.5 LOCAL RESOURCES AND INFRASTRUCTURE

5.5.1 Power Supply

The nearest power lines are located in the town of Yellow Pine, roughly 10 mi to the northwest. The power line to Yellow Pine would be insufficient to support a mining operation. Power lines would need to be installed / upgraded from the main regional Idaho Power Company (IPCo) substation at Lake Fork to the Project site. A description of the proposed power transmission line route is addressed later in Section 18 of this Report.

5.5.2 Water Supply

Midas Gold has four permanent and three temporary water rights in the district (collectively, "Water Rights"). The permanent Water Rights were transferred from the estate of J.J. Oberbillig and Bradley (Table 5.2). One of these Water Rights (77-7293) is currently shown in the IDWR database as owned by Bradley, however, Midas Gold has provided title information to IDWR, which has acknowledged that the documentation is sufficient, and is in the process of updating the database listing.

Table 5.2: Water Rights Summary

Water Right ID	Type	Source	Location	Beneficial Use	Maximum Diversion Rate (ft^3/s)	Maximum Annual Diversion (acre-feet)
77-7122	Surface Water	EFSFSR	NW ¼ of the NW ¼ , Section 14,T 18N, R9E	Storage and Mining	0.33	7.1
77-7141	Ground Water	Well	SW ¼ of the SW ¼, Section 11, T18N, R9E	Domestic	0.20	11.4
77-7285	Ground Water	Well	SE ¼ of the NE ¼, Section15, T18N, R9E	Storage and Mining	0.50	39.2
77-7293	Surface Water	Unnamed Stream (Hennessey Creek)	SW¼ of the NE¼, Section3, T18N, R9E	Mining	0.25	20.0
Source: IDWR, 2014						

Midas Gold's current water rights are insufficient to support the proposed Stibnite Gold Project development plan included herein, and additional rights will need to be secured through direct permit application and subsequent approval of such rights from the IDWR. Additional information regarding water rights and permitting are included in Section 20.

5.5.3 Rail

The Idaho Northern Pacific Railroad (INPR) is a Class II railroad that owns railroad tracks that terminate in Cascade, Idaho. The INPR operates the Cascade Branch rail line on approximately 100 mi of track between Payette, Idaho and Cascade with a switch yard in Emmett, Idaho. Track runs from Cascade to Payette connecting the line to the Union Pacific Railroad which is capable of reaching ports in California, Oregon, Washington and British Columbia. The INPR also operates a tourist train, the Thunder Mountain Line, on its Cascade Branch, which runs from Horseshoe Bend to Banks, Idaho.

Active freight service to Cascade ceased in the mid-1990s, but INPR continues to maintain the line to Cascade and conducts annual maintenance and inspections. INPR owns land at the terminus of the rail line for switching and transload facilities. Currently, facilities at the Cascade end of the track are limited.

Also serving the area and connecting to the Union Pacific Railroad is the Boise Valley Railroad (BVRR) at Nampa, Idaho located approximately 176 mi from the Project site. Currently the BVRR is a short-line railroad connecting Nampa with the state capital Boise, Idaho. Of the two rail lines, the BVRR is much further from the Project site; however, in May 2010, the City of Boise signed a letter of intent with the BVRR to explore construction of a trans-load and intermodal services facility in the southeast of Boise. Though construction of the proposed facility has not progressed beyond the initial letter of intent, when constructed, this facility would enable container freight to transfer directly from truck to train. Currently the nearest facility for direct container handling of the type proposed is in Portland, Oregon.

5.5.4 Ports

The closest access for sea transportation is through the ports of Portland, Oregon; Tacoma, Washington; Seattle, Washington; and Vancouver, British Columbia. Each of these ports is located in the Pacific Northwest and can be accessed by truck, or by rail with distances ranging from 573 to 727 mi from the Project. The Port of Portland is the closest of these four options; Terminal Six is the predominant container terminal at the port and is presently served by Hanjin Shipping on a weekly basis.

Additionally, The Port of Lewiston, Idaho, is located on the Clearwater River, just upstream from its confluence with the Snake River and is approximately 274 mi from the Project site. The port is served by truck and rail, and loads barges for shipment down the Snake and Columbia Rivers. The port is used primarily for shipping agricultural products. Wheat is shipped in bulk, but many of the other commodities are shipped in containers. The port also hosts a trans-loading facility where items are containerized for shipment. Containers travel down the Columbia to Portland’s Terminal Six a few days prior to being loaded onto a Hanjin vessel for Asian ports of call.

5.5.5 Communications

In 2013, Midas Gold completed a microwave relay tower atop a 9,000-ft peak on the east side of the property. The tower is on leased patented land and provides a reliable long-term link to the regional communications hub on Snowbank Mountain 52 mi to the southwest. The relay operates at 5.8 GHz and uses a 6 ft diameter parabolic antenna (40 ft above surface grade on the Stibnite end of the link) to provide a high bandwidth connection to a commercial leased tower facility, access to which is maintained year-round by the Federal Aviation Administration (FAA). A second smaller radio system relays the signal down to the valley floor via an intermediate tower near Midas Gold's Very High Frequency (VHF) repeater at West End. At the Stibnite tower sites, continuous and reliable power is provided by solar panels and battery systems designed to withstand the winter conditions at these locations.

Another 20 mi microwave link connects the Snowbank facility directly to Midas Gold’s Donnelly office, providing an entirely private and Midas Gold-owned communication path. A virtual private network connects the Boise office directly into this system and creates an environment where all Idaho facilities are under one virtual roof with respect to electronic data. Local servers are backed up off-site on a nightly basis to a Midas Gold-owned co-located server at the Syringa Networks data center adjacent to the Boise airport.

5.5.6 Potential Processing Site

The majority of the Project area is characterized by steeply-sloping, mountainous terrain. Flat terrain with competent foundation conditions suitable for mine infrastructure is generally limited; these areas are typically in the valley-bottoms, near the colluvium/alluvium/bedrock contact, which is consistent with infrastructure siting by previous mine-operators.

The process plant site selected to support the 2012 PEA was located on the southeast side of the confluence of Meadow Creek and the EFSFSR. While the PEA plant area is relatively flat and open, centrally located, with minimal underlying mineral resource potential, the area possesses pervasive wetlands, challenging foundation conditions, and is in very close proximity to sensitive jurisdictional waters. Following publication of the PEA, alternative process plant sites were identified by the Midas Gold team; consequently, a more comprehensive study was deemed appropriate to arrive at the preferred layout. The following methodology was used to arrive at the preferred process plant site:

1) Identify the primary physical constraints that limit the area that could be considered for process plant infrastructure such as: geotechnical constraints, avalanche constraints, regulatory constraints, project development constraints, etc.
2) Develop a scorecard that includes the key drivers/criteria that influence selection of the preferred process plant layout. The criteria could include: environmental, permitting and social considerations; safety considerations; capital expenditures (CAPEX); operating expenditures (OPEX); and operability considerations.
3) Develop conceptual project layouts that honor the preceding physical constraints with consideration to the key drivers.
4) Populate the scorecard in a workshop environment to identify the preferred process plant layout.

Following this process, a large, gently sloping area immediately northeast of the confluence of Meadow Creek and the EFSFSR was selected as the preferred processing plant location. Section 18 provides a detailed discussion on the layout of the process plant; a simplified version of the site layout is provided on Figure 5.2.

5.5.7 Potential Tailings Storage Area

Approximately 98 million tons of mineralized material are expected to be processed during the 12-year mine life of the Project as contemplated in this PFS. Ideally, from an environmental, technical and financial perspective, all of the tailings generated from the operation would be stored in a single storage facility. To determine the preferred location for the tailings storage facility (TSF), a siting assessment was completed that identified five locations that could provide sufficient storage capacity to contain the expected tailings quantities. An additional 14 smaller sites that could contain a portion of the required tailings storage in a second, separately managed facility, were also identified (SRK, 2012).

The preferred tailings site, based on considerations such as: topography, hydrology, use of previously disturbed areas, environmental management and closure considerations, proximity to the processing plant, and expected cost, was determined to be in the Meadow Creek Valley. The valley has sufficient capacity for both tailings and waste rock, and a significant portion of the area has been previously disturbed by historical mining operations. This site was identified and incorporated into the PEA and continues to be the preferred tailings storage site since it, among other things, keeps incremental disturbance to a minimum by overlapping on pre-existing historically disturbed areas used previously for tailings disposal. A comprehensive description of the Meadow Creek TSF is provided in Section 18; a simplified figure showing the location of the TSF is presented on Figure 5.2.

Some of the land in the Meadow Creek Valley is owned by Midas Gold and comprises patented mining claims; the balance of the land in the valley is Federal land and is controlled by the USFS.

5.5.8 Potential Waste Rock Disposal Area

There are several locations on the Stibnite Gold Project site where uneconomic mineralized material or waste rock could be stored, which were evaluated in a similar manner and with similar considerations as the siting of the TSF. The preferred storage area for the Yellow Pine and Hangar Flats waste rock is in the Meadow Creek Valley downstream of the TSF that would, in addition, provide a robust geotechnical stability buttress for the TSF. This site is preferred since it, among other things, keeps incremental disturbance to a minimum by overlapping on pre-existing historically disturbed areas used previously for tailings disposal and spent heap leach ore disposal, and keeps the waste rock and tailings within the same area. The preferred storage location for the West End waste rock is above the existing West End waste rock storage facility and, mostly, in the mined-out Yellow Pine open pit, which would enable the EFSFSR to be reestablished to its original gradient, facilitating long-term fish passage to the headwaters of the EFSRSR and Meadow Creek. Some of the proposed waste rock storage land is owned by Midas Gold and comprises patented mining claims; the rest of the land in the valley is Federal land and is controlled by the USFS. This layout keeps the maximum amount of disturbance within the existing footprint of historical disturbance. Sections 16 and 18 provide additional details on the waste rock storage areas; a simplified layout is provided on Figure 5.2.

5.5.9 Labor

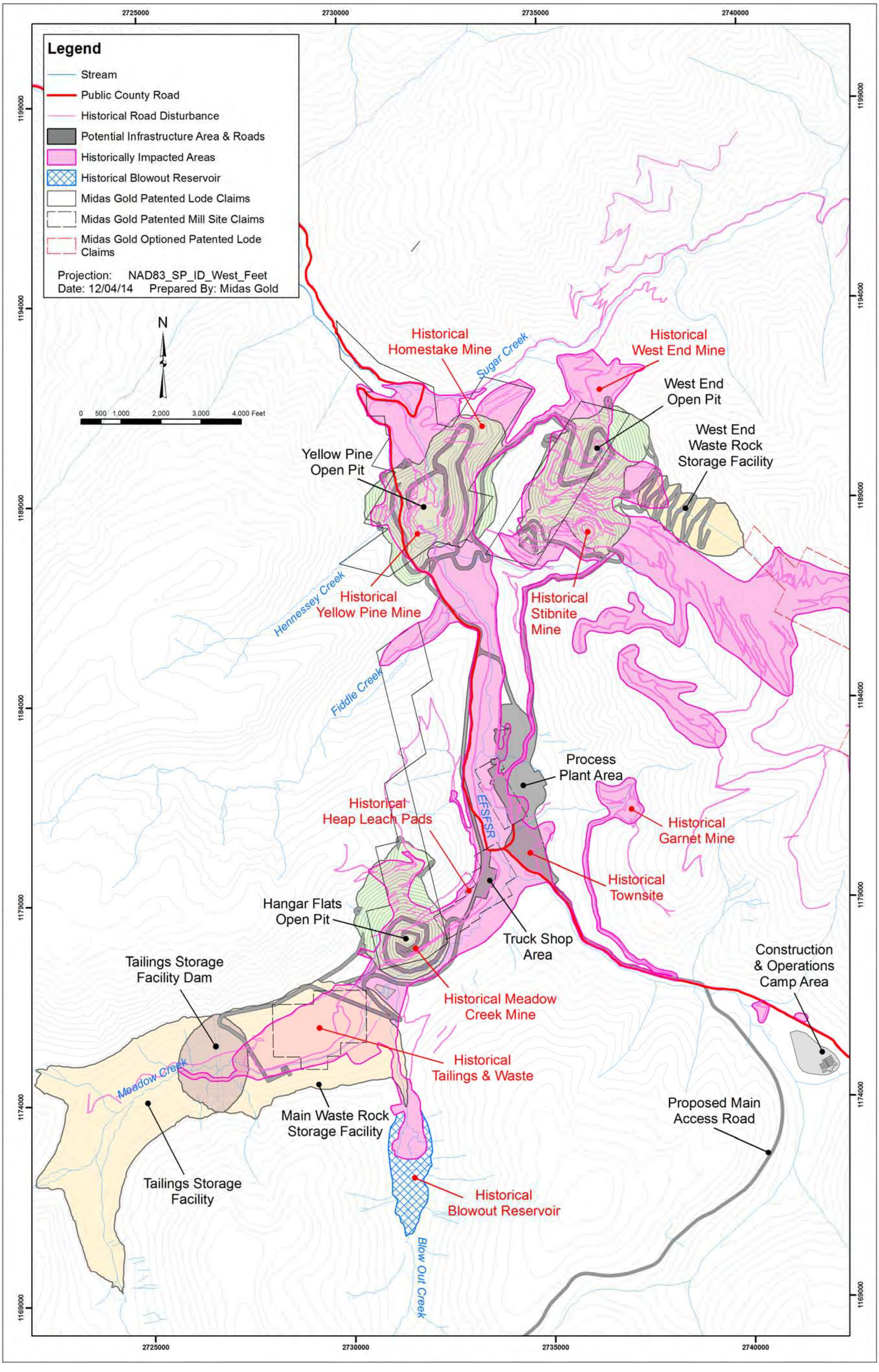
Yellow Pine, which is the nearest town, is located approximately 14 mi to the west of the Project. It has a population of approximately 60 people during the summer months, up to 40 in the winter, and limited services such as a general store, a restaurant, and a few lodging facilities. The nearby Valley County towns of McCall, Donnelly and Cascade, and surrounding areas have a combined population of several thousand people with many diverse services available.

Skilled miners and mining professionals, as well as local laborers and equipment operators, would be identified from within Valley County and adjacent Adams County, where feasible, with additional workers sourced throughout Idaho if necessary.

Based on the currently envisioned Project, Midas Gold would likely become the largest employer in Valley County and Adams County, paying higher salaries than any other industry except the federal government. These two counties have some of the highest unemployment rates in Idaho, which is nearly double the rate in Boise. Midas Gold jobs would revitalize the local manufacturing sector and provide an important complement to the region's recreation industry. In addition to the long-term operations-related employment opportunities, Midas Gold would also employ a large number of construction workers during the construction phase, which would bolster the slumping real estate industry. The property and sales taxes generated from the mining operations would help support the region's schools and infrastructure, which have been under recent economic stress. The infusion of new economic activity would likely help support every industry in the regional economy.

Additional details on the Project labor requirements and approaches to meeting those needs are discussed in Section 20.

Figure 5.2: General Site Layout



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6 HISTORY

6.1 OWNERSHIP AND ROYALTIES

During the first half of the 20th century, two major landowners were working the Stibnite Mining District. The eastern part was partially consolidated by United Mercury Mines, whereas the western part was controlled by Bradley. Bradley production was initially from the underground Meadow Creek mine (c. 1927 to 1937) and later from the larger Yellow Pine underground and subsequently open pit mine (1937 to 1952). Bradley's consolidation of the western district led to the Oberbillig family receiving royalties on some of the claims mined by Bradley. Mining claims associated with the Meadow Creek Mine and Yellow Pine Mine (first staked in 1914 and 1923, respectively) were patented during this period. Bradley operated Yellow Pine until 1952 and optioned the nearby Cinnabar mine in the late 1950s. Mining operations ceased after a worldwide collapse in antimony and mercury prices following the end of the Korean War, while milling and smelting continued from stockpiled ores, as well as antimony-bearing materials from the Coeur d'Alene district, as well as tungsten ores from the Springfield and Ima tungsten mines. The former mill and smelter were subsequently dismantled and the Stibnite town site abandoned with many of the cabins and other buildings comprising the town site and other facilities moved elsewhere.

Aside from minor mining and processing of stibnite ores from the Stibnite District and mercury mining at the nearby Cinnabar mine in the 1960s, the Stibnite/Yellow Pine district lay dormant until the early 1970s, when a sharp rise in gold prices and the advent of heap-leach processing technology for oxide gold ores revitalized exploration in the District. Operators who conducted exploration and/or minerals extraction during this era included, in chronological order, Louisiana Land and Exploration Company, Canadian Superior Mining (U.S.) Ltd. (**Superior**), El Paso Mining and Milling (**El Paso**), Rancher's Exploration Company (**Ranchers**), Twin Rivers Exploration, MinVen Corporation (**MinVen**), Pioneer Metals Corporation (**Pioneer**), Hecla, Barrick Gold Corporation (**Barrick**, and formerly American Barrick Resources), and SMI.

Hecla delineated a small oxide resource at the Hangar Flats Deposit, but focused mainly on mining the nearby Homestake oxide gold deposit, which overlies the northeastern portion of the Yellow Pine Deposit. Superior delineated much of what is now the West End Deposit and they brought that area into production in 1982. Superior was ultimately acquired by the Superior Oil Company of Houston, Texas, which, in turn, was acquired by Mobil Oil. Mobil sold the West End Mine in 1986 to a 50/50 joint venture of Pioneer and MinVen, both small Canadian-registered companies. Pioneer was the mine operator until it experienced financial problems in 1990, and ownership was conveyed to SMI, owned primarily by MinVen. MinVen later experienced financial problems and the mine was conveyed to Dakota Mining Company (**Dakota**). Operations in the district ceased after the 1997 season, when Dakota merged with USMX Inc. Rapidly falling gold prices in 1997, internal company financial problems, increasing environmental and regulatory issues, and delays in obtaining necessary operating permits led to the mine closure.

In 1990, during the course of these operations, six lode claims and 30 mill site claims (including mineral rights) were patented with ownership going to the Oberbillig Family Estate. These Oberbillig Estate patented lands and the 5% NSR royalty interest on the Bradley Estate are currently the subject of purchase option agreements with Midas Gold (both Promissory Notes mature June 2, 2015). Midas Gold will purchase the royalty which would, in effect, have Midas Gold pay the royalty to itself.

On June 2, 2003, Vista's wholly owned subsidiary Vista Gold US Inc. (**Vista US**) entered into an Option to Purchase Agreement with Bradley regarding 17 patented lode mining claims owned by Bradley that covered the majority of the Yellow Pine Deposit. In addition, Vista, through its wholly owned affiliate, Idaho Gold Resources, LLC (**IGR**), acquired eight unpatented lode mining claims, also in the Yellow Pine Deposit area. On February 22, 2011, MGI entered into a combination agreement with Vista US and IGR whereby these entities became wholly owned subsidiaries of Midas Gold. Midas Gold made final payment under the Option to Purchase on November 28, 2012 and now holds the title to these claims.

In 2006, much of the western portion of the district was staked by Niagara Mining and Development, a subsidiary of Gold Crest Mines Inc. (Gold Crest). These unpatented claims surround the patented lands of both the Bradley and Oberbillig Family estates. Additional, unpatented claims were staked by Gold Crest in 2007 covering the eastern portions of the district. All Gold Crest claims were purchased by MGI in 2009, and agreements were negotiated with the patented landowners.

On April 28, 2011, MGI's wholly owned subsidiary, MGI Acquisition Corp. (MGIAC), entered into an agreement with the owners of the six Fern patented mineral claims and now owns those rights 100%.

On May 1, 2011, MGI's wholly owned subsidiary, MGIAC, entered into an option agreement with the owners of a number of patented and unpatented mineral claims comprising the former Cinnabar Mine property JJO, LLC a limited liability company and the personal representative of the estate of J.J. Oberbillig, whereby MGIAC had the right but not the obligation to acquire these claims over a period extending to May 1, 2017 in exchange for certain payments. MGIAC has made all payments required to date under the option agreement and the option agreement remains in effect.

MGI subsequently completed staking of additional claims and corrected claim deficiencies between 2009 and 2012.

The entire property (excluding the Cinnabar group of claims) is subject to the May 9, 2013 1.7% gold only NSR royalty held by Franco-Nevada Corporation. Midas Gold's subsidiaries have a one-time right to repurchase one third of the royalty for US$9 million before May 9, 2016 thereby reducing the royalty to 1.13%.

6.2 PAST EXPLORATION AND DEVELOPMENT

There have been two major periods of exploration and development operations in the District prior to Midas Gold gaining control, one spanning from the early 1900s through the 1950s and another during the period from the early 1970s through the mid-1990s. The history of development and mining in the district is summarized in numerous publications and additional references therein including: Larsen and Livingston (1920); Schrader and Ross (1926); White (1940); Cooper (1951); Hart (1979); Waite (1996); and Mitchell (1995; 2000) and various unpublished reports and documents. Much of the information contained in the text below is taken from these published sources and from unpublished company records.

The mining history of the region began in 1894 when the Caswell brothers began a sluice box operation in Monumental Creek in what is now known as the Thunder Mountain Mining District, located east of Stibnite. By 1902 a gold rush was underway to the Thunder Mountain District with associated development of roads and creation of the town of Roosevelt. By 1909, the gold rush was essentially over; that spring, a mudslide blocked Monument Creek creating present-day Roosevelt Lake and submerging the town of Roosevelt. During the Thunder Mountain gold rush, many prospectors passed through the area now known as the Stibnite-Yellow Pine District, discovering mercury, antimony, silver and gold. However, no work of any significance was completed until around 1917, when the World War I demand for mercury led to the development of several properties east of the main Project area, including the Hermes group of claims located by Pringle Smith in 1902, and the Fern group located by E. H. VanMeter in 1917 (Larsen and Livingston, 1920; Schrader and Ross, 1926).

Between the 1920s and late 1990s, numerous prospects were discovered and explored using soil sampling, rock sampling, trenching, drilling, geophysical methods and geology. Several of these prospects were developed into successful mining operations. Production records for these operations are discussed in Section 6.4. The history of exploration and development of the major deposits is discussed below and the major exploration activities by past operators and Midas Gold are summarized in Section 9.

6.2.1 Hangar Flats Deposit

Gold and antimony mineralization were discovered in the Hangar Flats area around 1900. Albert Hennessy staked the first claims here in 1914. Initial prospecting and development attempts focused on outcropping gold-silver-antimony mineralization, principally in the Meadow Creek area. By the mid-1920s, Albert Hennessy and his partners, who included J.J. Oberbillig, had established the Meadow Creek Silver Mines Company (MCSM) and had carried out intermittent, but considerable underground development work on what became known as the Meadow Creek Mine. Homestake Mining Company (Homestake) optioned the property and conducted sampling and metallurgical investigations during this period, but decided not to complete a purchase of the property after initial metallurgical investigations indicated that they were unable to process the complex gold-antimony ores (Mitchell, 2000). In 1921, MCSM was superseded by United Mercury Mines and, by the mid-1920s, the Meadow Creek Mine area was consolidated under Bradley interests, and the mine was systematically explored and developed on six levels with numerous drifts, crosscuts, raises, winzes, and stopes. It subsequently produced gold, silver, and antimony from sulfide ores, which were milled on site from 1928 through 1938. Mine workings were systematically mapped and sampled, and modern style exploration core drilling (from both the surface and underground) was carried out to guide the mine development. About 25,426 ft of underground workings were developed in the Meadow Creek Mine, while substantial additional core drilling was completed during this period (for details of drilling during this time period reference section 10 of this Report). The Meadow Creek Mine produced gold, silver, and significant quantities of antimony between 1928 and 1937. Figure 6.1 shows the processing facility and tailings pond for the Meadow Creek Mine during this time period. Most of the historic underground maps, tunnel assays, drill logs, and drill assay results can be found in Midas Gold's files or the Idaho Geological Survey archives.

In 1937, the Meadow Creek Mine was shut down and production shifted to development of the Yellow Pine deposit in 1938. Beginning in 1943, a mostly unsuccessful attempt was made to re-open portions of the old Meadow Creek Mine workings to explore for antimony and tungsten in support of the war effort. From 1943 to 1945 additional core drilling was completed in the mine, all post-operations. A small amount of tungsten mineralized material was reportedly mined during this period from two levels of the mine that were not caved or flooded (Cooper, 1951).

Figure 6.1: Bradley Mining Company Processing Plant and Tailings Pond

Source: Photograph circa 1942, courtesy of Robin McRae

From 1951 through 1954, the Defense Minerals Exploration Administration (DMEA) carried out an underground exploration program immediately to the north of the Meadow Creek Mine. The impetus for that work was provided by the Defense Production Act of 1950 (cf. 15 CFR §§700 to 700.93). It provided monetary assistance for companies to locate new reserves of strategic and critical minerals (Mitchell, 2000). If mineralized material was discovered, the companies that received assistance were required to reimburse the government from the proceeds of the operation. If no economic mineralization was discovered, the government loans were forgiven. Through the DMEA program, Bradley developed approximately 4,900 ft of underground workings on three levels (Mitchell, 2000) in the area immediately north of the Hangar Flats Deposit. Systematic mapping and sampling of the workings were carried out with the mining of bulk samples that were collected at roughly 5 to 10 ft intervals. Drilling from underground stations was also carried out. Detailed drill logs and systematic assaying were well documented.

In the late 1970s, Ranchers acquired interests in the district from Bradley and completed a large soil grid over the trace of the Meadow Creek Fault system, including the area adjacent to the old Meadow Creek Mine. Ranchers' work outlined a number of large gold-in-soil anomalies over the old mine site, along the trace of the Meadow Creek Fault system, and north several kilometres to the Yellow Pine Deposit. Ranchers completed some trenching, but no drilling on the anomalies in this area; instead they focused their work on the Yellow Pine and Homestake deposits (Mitchell, 2000).

In the late 1980s, Hecla acquired Ranchers' interests and conducted trenching and ground geophysical surveys, as well as drilling 27 shallow reverse circulation (RC) holes in the area of the historic Meadow Creek Mine. Their trenching and RC drilling outlined a broad, but ill-defined zone of gold mineralization above the old workings and along strike to the north, as well as under the old Meadow Creek mill and smelter complex along the base of the hill (where the old Meadow Creek adits were located). Subsequently, Hecla constructed a heap-leach pad over a portion of the main mineralized area due to the need to find a location to leach the oxide ores from the Homestake area of the Yellow Pine Deposit. No further work, other than reclamation of the heap by Hecla and the mill and smelter by government agencies, occurred until Midas Gold's work was initiated in 2009.

6.2.2 Yellow Pine Deposit

The first claims were staked in the Yellow Pine Deposit by prospector Al Hennessy in 1923 who, with J. L. Niday, formed the Great Northern Mines Company. In 1929, the claims were optioned to F. W. Bradley's Yellow Pine Mining Company which drove the Monday and Cinnabar tunnels on opposing sides of the valley. In 1933, these claims were sold to J.J. Oberbillig. By 1938, when the Meadow Creek Mine was shut down, exploration, development, and production shifted to the Yellow Pine Deposit (Mitchell, 2000). A substantial amount of drilling in this area was completed by numerous operators from the late 1930s through the 1990s.

Between 1933 and 1952, Bradley and the United States Bureau of Mines (USBM) completed systematic exploration and development drilling in the Yellow Pine and Homestake areas in several drilling campaigns. These drilling programs were spurred on by both the demand for antimony, after the U.S. Government declared antimony a strategic metal (The Strategic Minerals Act of 1939), and the discovery of significant tungsten by U.S. Geological Survey (USGS) geologist Donald E. White who was studying USBM drill core from the district in 1941. Subsequent exploration and development included both underground and open pit exploration and development drilling, mapping, sampling and mining. Figure 6.2 shows the Yellow Pine Open Pit in the early 1950s. During the World War II era, the Yellow Pine Mine was the major source of antimony and tungsten for the war effort and exploration during this period was focused on those commodities (Mitchell, 2000).

After operations shut down in 1952, little work was completed until the 1970s, when Ranchers and, later, its successor Hecla conducted extensive drilling campaigns on the deposit starting in the 1970s and continuing through the mid-1990s along with trenching, pit mapping, engineering, and environmental and metallurgical studies. Hecla completed a prefeasibility study of the project in 1987. Barrick optioned the property in the early 1990s in a joint venture with Hecla and completed additional drilling and metallurgical test work before dropping the option. Hecla relinquished its control of the property back to the Bradley estate interests after closure and reclamation of the oxide operations at the Homestake pit in the late 1990s (Mitchell, 2000). Vista completed an independent mineral resource estimate prepared in 2003 (Pincock, Allen and Holt, 2003) and a Preliminary Assessment by the same group in 2006 (Pincock, Allen and Holt, 2006) but conducted no work on site in support of these reports. No additional exploration or development work was completed until MGI acquired their interests by purchasing IGR in 2011.

Figure 6.2: Bradley Mining Company Open Pit Mine

Source: Photograph circa 1942, courtesy of Robin McRae

6.2.3 West End Deposit

Gold mineralization was first discovered along the West End Fault by Bradley interests in the late 1930s or early 1940s; during this time Bradley's exploration focused on replacement of reserves at their Yellow Pine mining operation. Subsequent work sponsored by the USGS outlined a large biogeochemical and soil anomaly (Leonard, 1973) that led to systematic follow-up by Superior and its successors. A modern era of exploration and development stretched from the mid-1970s to the mid-1990s, prompted primarily by the rise in gold prices and the development heap-leach oxide gold recovery methods (Mitchell, 2000).

Superior conducted geological, geophysical, and geochemical investigations from 1974 to 1977 to evaluate the potential for heap-leach oxide gold in the West End and adjacent Stibnite deposit (now collectively known as West End). In 1979, Superior Oil Company, Superior's parent company, purchased Superior's outstanding shares and became sole owner of the West End Deposit. After completion of a favorable Environmental Impact Statement, five heap-leach pads were constructed, and a 2,000 - 3,000 st/d oxide mining operation began in 1982 (Figure 6.3). Open pit mining at the West End Mine and heap-leach processing was conducted by Superior until 1984 when ownership of the deposit once again changed hands when Mobil Oil purchased Superior Oil. The West End mine did not operate in 1985, however heap leach processing of previously mined material continued throughout 1985 (Mitchell, 2000).

Figure 6.3: Canadian Superior Mining (U.S.) Ltd. Heap Leach Processing Facility

Source: Photograph circa 1985, courtesy of the U.S. Forest Service

In 1986, Pioneer purchased the mine from Mobil with financing assistance from The Mining Finance Corporation and Twin Rivers Minerals who owned 25% of the West End Pit, and 18% of Pioneer's stock (Mitchell, 2000). At this time, Pioneer became the operator of the West End mine and continued to explore and produce until 1991. From 1991, ownership of the West End open pit mine and processing facilities changed hands from Pioneer to Pegasus Gold Corporation (Pegasus), and then to MinVen (later changed to Dakota). During this time the mining and exploration activities in the area continued under MinVen's (later Dakota's) subsidiary company, SMI. SMI continued to conduct sporadic drilling and development of the West End pit, including a small area on the east side of the West End Deposit known as the Stibnite pit, and a small pit approximately 1.5 miles to the south east known as the Garnet Pit, into the mid- to late-1990s. Between 1982 and 1994 crushed oxide material from the West End pits was placed in the Upper Meadow Creek Valley after being leached, neutralized, and rinsed (Mitchell, 2000) in an area now commonly referred to as the Spent Ore Disposal Area (SODA). For estimated production records during this time period see Table 6.3.

6.3 HISTORICAL MINERAL RESOURCE AND RESERVE ESTIMATES

Through the years after historic mining ceased in the 1950s, various companies have completed mineral resource estimates of all or portions of the Meadow Creek Mine (now called Hangar Flats), West End, and Yellow

Pine/Homestake deposits using different gold prices, cut-off grades, estimation methods, and datasets. These include multiple estimates by Ranchers, Hecla, Santa Fe Pacific Gold Corporation, Newmont Mining Corporation, and Barrick. These estimates are available in Midas Gold files, but were completed prior to 1995 and were not prepared in accordance with the requirements of Sections 1.2 and 1.3 of NI 43-101. There are no historic Mineral Resource or Mineral Reserve estimates that compare with the Mineral Resource estimates of this Report. Historic data files contain various estimations of oxide and sulfide mineralized material consisting of individual mineralized lenses within the Hangar Flats, West End, and Yellow Pine deposit areas, but the mineral resource estimates and supporting backup data are incomplete or were for only small portions of larger deposits and are, therefore, not pertinent and are not reported here.

In 2003, Vista contracted with Pincock, Allen and Holt (PAH) to complete an NI 43-101-compliant Mineral Resource estimate and Technical Report (Pincock, Allen, and Holt, 2003) on the Yellow Pine Deposit. This report was completed prior to any drilling by Vista or Midas Gold and has since been determined to be obsolete. The reader is referred to this report on the Canadian Securities Administrator's (CSA) System for Electronic Document Analysis and Retrieval (SEDAR) for details of the PAH resource estimation procedures and results.

Midas Gold has completed several Mineral Resource estimates for the Project. These include a maiden Mineral Resource estimate for the Hangar Flats, West End and Yellow Pine deposits (SRK, 2011) followed by updated Mineral Resource estimates described in the PEA (SRK, 2012). The reader is referred to these reports by the issuer on SEDAR for details on procedures, assumptions, caveats and results from the previous Mineral Resource estimates. Information in this Report supersedes information reported in the PAH report, the SRK 2011 report and the PEA.

6.4 HISTORICAL PRODUCTION

Historical production figures, because of limited surviving records, are estimates that have been pieced together from several sources. Victoria E. Mitchell of the Idaho Geological Survey (IGS) published a detailed report in 2000 titled "History of the Stibnite Mining Area, Valley County Idaho" and much of the history and production numbers used in this Report come from that document. Mitchell's report however, does not detail all of the production from the three deposits for all of the years that their respective mines operated and, as a result, other sources were utilized to fill in the gaps. Sources include public filing reports from the US Securities and Exchange Commission (SEC), unpublished company production records, Idaho State Mine Inspection records, and USBM reports. Occasionally, these sources contained conflicting data, in which case the company's production records were utilized. The production figures in many instances are only estimates and are not reported consistently for gold, silver, and antimony.

Table 6.1 summarizes production for the Project by area, while additional details are provided below.

Table 6.1: Stibnite District Estimated Historical Production

Area	Production Years	Tons Mined (st)	Recovered Au (oz)	Recovered Ag (oz)	Recovered Sb (st)	Recovered WO_3 (units)[1]
Hangar Flats	1928 - 38	303,853	51,610	181,863	3,758	67
Yellow Pine	1938 - 92	6,493,838	479,517	1,756,928	40,257	856,189
West End	1978 - 97	8,156,942	454,475	149,760	-	-
Totals		14,954,633	985,602	2,088,551	44,015	856,256

Note:
1. *A unit of WO_3 (tungsten trioxide) is 1% of a short ton (20 pounds), and WO_3 is 79.3% tungsten. A short ton unit of WO_3, therefore, equals 20 pounds of WO_3 and contains 15.86 pounds of tungsten.*

6.4.1 Hangar Flats Deposit

Gold, silver, antimony, and tungsten were produced from the Hangar Flats Deposit from 1928 to 1938. Based on available compiled records, the totals listed in Table 6.2 provide an approximation of the production from underground operations in the Meadow Creek Mine.

Table 6.2: Hangar Flats Deposit Estimated Production Records

Company Name	Production Year	Tons Mined (st)	Recovered Au (oz)	Recovered Ag (oz)	Recovered Sb (st)	Recovered WO_3 (units)[1]
Bradley	1928-31	19,767	Unknown	Unknown	Unknown	-
Bradley	1932	34,366	6,916	18,488	489	-
Bradley	1933	45,710	10,412	29,817	588	-
Bradley	1934	54,000	10,491	25,384	404	-
Bradley	1935	50,965	8,373	25,217	550	-
Bradley	1936	43,324	7,798	32,615	729	-
Bradley	1937	39,521	5,514	36,572	755	-
Bradley	1938	16,200	2,106	13,770	243	67
TOTAL		303,853	51,610	181,863	3,758	67

Notes:
1. *A unit of WO_3 (tungsten trioxide) is 1% of a short ton (20 pounds), and WO_3 is 79.3% tungsten. A short ton unit of WO_3, therefore, equals 20 pounds of WO_3 and contains 15.86 pounds of tungsten.*

6.4.2 Yellow Pine Deposit

Gold, silver and antimony were produced from the Yellow Pine Deposit starting in 1938, with the addition of tungsten in 1941 with continuous production from 1938 to 1952. Based on available compiled records, the totals listed in Table 6.3 provide an approximation of the production from underground and open pit operations during this time period. Additionally, from 1989 to 1992 gold was produced from open pit operations in the Homestake Mine, an oxide gold deposit which overlies the northeastern portion of the Yellow Pine Deposit.

Table 6.3: Yellow Pine Deposit Estimated Production Records

Company Name	Production Year	Tons Mined (st)	Recovered Au (oz)	Recovered Ag (oz)	Recovered Sb (st)	Recovered WO_3 (Units)[2]
Bradley	1938	22,680	1,423	3,917	136	-
Bradley	1939	56,074	5,810	14,844	228	-
Bradley	1940	132,297	12,401	15,825	18	-
Bradley	1941	95,156	10,355	18,981	380	27,921
Bradley	1942	96,861	2,714	85,161	2,801	181,230
Bradley	1943	178,747	4,529	109,307	2,734	303,502
Bradley	1944	211,382	6,110	74,498	2,031	233,664
Bradley	1945	109,796	6,505	87,815	2,895	85,572
Bradley	1946	147,505	14,276	68,564	1,477	-
Bradley	1947	584,483	44,393	324,582	6,699	-
Bradley	1948	655,682	49,400	318,090	7,948	-
Bradley	1949	610,988	68,423	127,403	2,104	-
Bradley	1950	620,800	61,763	177,594	3,747	5,899
Bradley	1951	546,163	39,242	226,274	4,575	11,220

Company Name	Production Year	Tons Mined (st)	Recovered Au (oz)	Recovered Ag (oz)	Recovered Sb (st)	Recovered WO_3 (Units)[2]
Bradley	1951[1]	26,355	-	-	-	4,990
Bradley	1952	310,201	24,747	104,073	2,484	2,191
Hecla	1988	278,193	20,701	-	-	-
Hecla	1989	910,475	29,436	-	-	-
Hecla	1990	900,000	57,747	-	-	-
Hecla	1991	Unknown	17,542	-	-	-
Hecla	1992	Unknown	2,000	-	-	-
TOTAL		6,493,838	479,517	1,756,928	40,257	856,189

Notes:
1. *Re-processing tailings.*
2. *A unit of WO_3 (tungsten trioxide) is 1% of a short ton (20 pounds), and WO_3 is 79.3% tungsten. A short ton unit of WO_3, therefore, equals 20 pounds of WO_3 and contains 15.86 pounds of tungsten.*

6.4.3 West End Deposit

Gold and silver were produced from the West End Deposit from 1982 to 1993. Based on public filings, published reports and unpublished company production records, the totals listed in Table 6.4 provide an approximation of the production from operations in the West End, Splay, Stibnite and Garnet pits, all of which, except Garnet, are located within the West End Deposit.

Table 6.4: West End Deposit Estimated Production Records

Company Name	Production Year	Tons Mined (st)	Recovered Au (oz)	Recovered Ag (oz)	Recovered Sb (st)	Recovered WO3 (Units)
Superior	1978	1,500	60	-	-	-
Superior	1982	200,000	7,832	3,287	-	-
Superior	1983	480,000	29,000	8,207	-	-
Superior	1984	487,295	28,645	8,107	-	-
Superior	1985	-	-	-	-	-
Superior	1986	630,865	45,508	28,719	-	-
Superior	1987	764,121	40,802	25,750	-	-
Pioneer	1988	278,193	32,347	17,418	-	-
Pioneer	1989	910,475	29,436	9,778	-	-
Pioneer	1990	982,240	63,357	9,942	-	-
Pioneer	1991	863,783	31,555	11,008	-	-
Pioneer-Pegasus	1992	950,000	31,549	12,818	-	-
MinVen-Dakota	1993	91,000	2,042	1,330	-	-
SMI	1994	-	-	-	-	-
SMI	1995	300,340	20,949	5,378	-	-
SMI (In Garnet Creek Pit)	1995	300,130	59,190	-	-	-
SMI	1996	927,000	32,203	8,019	-	-
Total		8,166,942	454,475	149,760	-	-

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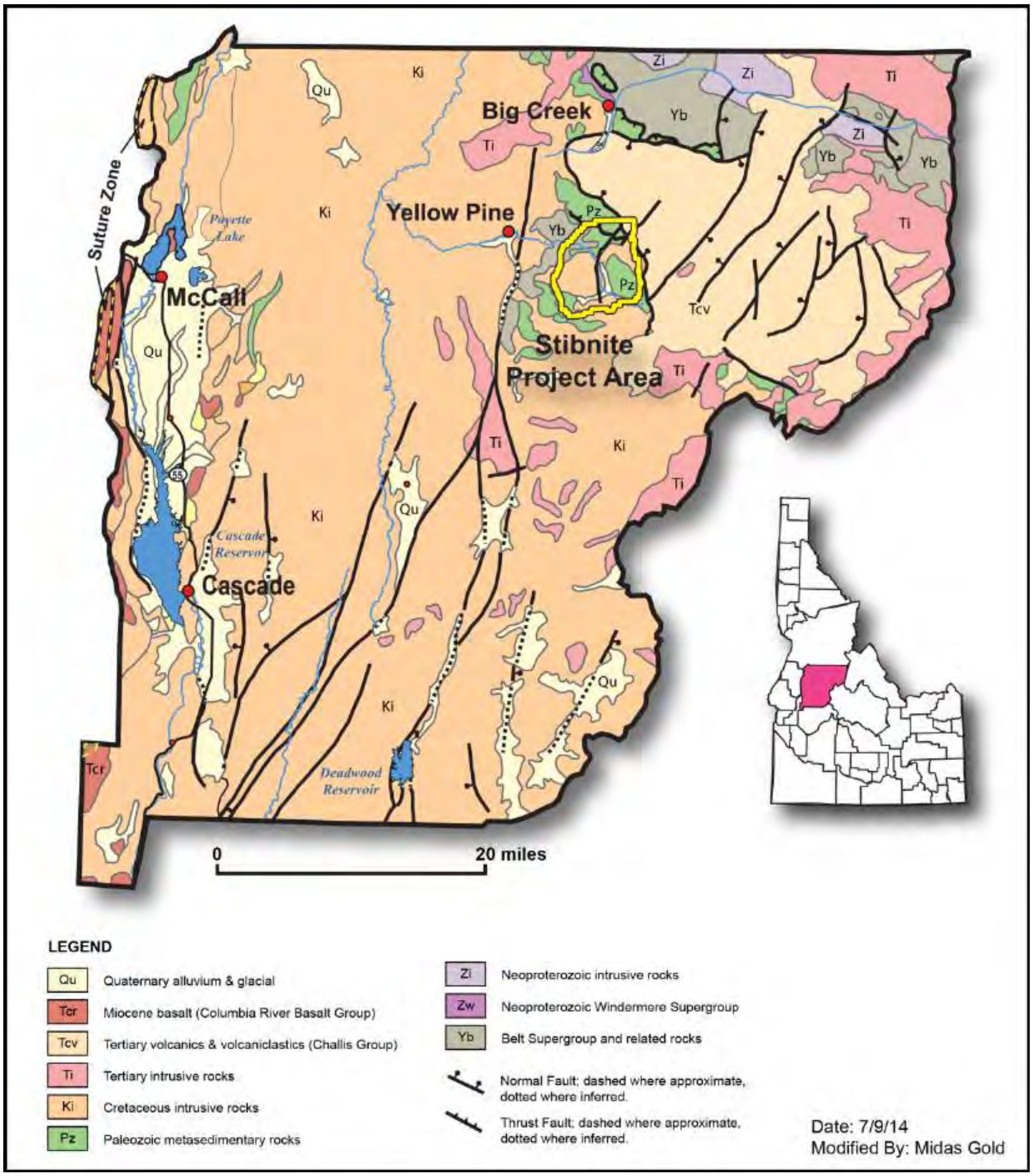
7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Project area is located in the Salmon River Mountains, a high-relief mountainous physiographic province in central Idaho. Bedrock in the region can be subdivided into several groups based on age, lithology and stratigraphic relationships. In a broad sense, rock sequences in the region can be subdivided into rocks that are part of the pre-Cretaceous "basement," the Cretaceous Idaho Batholith, Tertiary intrusions and volcanics, and younger unconsolidated sediments derived from erosion of the older sequences and glacial materials (Figure 7.1).

Figure 7.1: Valley County, Idaho Regional Geologic Map



Source: Modified from Lewis, 2002

The pre-Cretaceous basement rocks record the development and subsequent tectonic overprinting of the western Laurentian continental margin which formed during a protracted rifting event from Neoproterozoic through middle Paleozoic time. This rifting event was accompanied by deposition of rift and passive margin sediments along the western edge of ancestral North America. Poorly preserved remnants of the rift and subsequent passive margin sedimentary sequences are exposed in the region as discontinuous roof pendants in a broad northwesterly trending belt adjacent to or as inliers within the Idaho Batholith extending from southeast Idaho to at least as far north as northeast Washington and beyond (Lund, et al., 2003; Lewis, et al., 2012).

These rocks record a long and varied sedimentary record spanning Proterozoic through Paleozoic time and likely correlate with the Mesoproterozoic Belt Supergroup, the Neoproterozoic Windermere Supergroup and the Neoproterozoic to lower to middle Paleozoic passive margin miogeoclinal successions (Lund, et al., 2003; Lewis, et al., 2012). Subsequent metamorphism, structural complexity and preservation of only small erosional remnants of these sequences make an accurate measurement of original thicknesses, stratigraphic associations and original facies relationships difficult. However, recent regional mapping in conjunction with more definitive age determinations using high resolution detrital zircon dating methodologies suggests the youngest metasedimentary rocks within the Project area are correlative in part to rocks exposed in southeast Idaho in the Bayhorse region and in the northwestern Panhandle of Idaho and record the Neoproterozoic rifting event and development of a passive continental margin (Lewis, et al., 2014). This mapping and dating is being conducted by the Idaho Geologic Survey (IGS). Research is ongoing and findings will be published when necessary and appropriate by the researchers involved. Mapping and dating reported here are the results of this ongoing research.

Although difficult to document at the local scale, regional mapping indicates pre-Cretaceous basement rocks in the region underwent several periods of deformation, likely including the Cretaceous-Tertiary Sevier and Laramide orogenies. Each subsequent orogeny resulted in progressively eastward contraction of the miogeoclinal sequence and underlying, older rift-related units. The Salmon River Suture Zone, situated west of the Project area (Figure 7.1), marks the transition zone between Precambrian continental crust of North American affinity to the east and accreted Neoproterozoic to Paleozoic oceanic crust to the west, as defined by various petrologic and geochemical studies as well as isotope values and geophysical models (Piccoli and Hyndman, 1985; Kleinkopf, 1988; Lund and Snee, 1988; Strayer, et al., 1989).

After rifting and development of the passive margin, regional folding and faulting in the early Paleozoic was followed by extensive early Mesozoic folding, extensive west to east thrust faulting in the middle and late Mesozoic, and late Mesozoic normal faulting (Lund, et al., 2003). The Idaho Batholith intruded the sedimentary sequences in mid-to-late Cretaceous. The western margin of the Idaho Batholith is metamorphosed and foliated parallel to the Salmon River Suture Zone, which indicates that it was emplaced while the suture zone was still active (Manduca, et al., 1993). The eastern margin is overprinted by younger Tertiary caldera complexes (Fisher, et al., 1992). Intrusive activity and volcanism continued through the Tertiary during uplift as the batholith was unroofed. During the Eocene the Challis volcanics blanketed the region to the east. Eocene, and later Miocene, Basin and Range normal faulting reactivated pre-existing Cretaceous structures resulting in a series of normal fault-bounded basins. To the west of the Project area, evidence of widespread extensional deformation is concentrated in the Late Cretaceous Western Idaho Shear Zone, resulting in the development of the Long Valley basin near the towns of New Meadows, McCall, Donnelly and Cascade. The area affected by the Western Idaho Shear Zone displays two orientations of steep faults: one set of normal faults strikes north-south and is parallel to fabrics within the suture and the other sets strike east-west and northeast and accommodate components of both normal and strike-slip movement. Similar structural trends are evident in the area surrounding the Project. Approximately 10 miles to the west of the Project area, the mile-wide, 80 mile-long north-south trending Johnson Creek-Profile Gap Shear Zone is marked by dike swarms, heavy fracturing, multi-stage brecciation and pervasive alteration, and shows evidence of both Cretaceous and Tertiary intrusive and tectonic activity. The Meadow Creek Fault Zone (MCFZ), parallel to the Johnson Creek Profile Gap structure, is situated along the west side of the Thunder Mountain Caldera and can be traced for over 10 miles in a north-south direction and has similar characteristics to the Johnson Creek Profile Gap structure.

Regionally, the Atlanta Lobe of the Idaho Batholith shows a progression from early mantle-derived metaluminous magmatism from 98 mega-annum (Ma [or million years]) to 87 Ma, followed by more voluminous crustal-contaminated peraluminous magmatism from 83 Ma to 67 Ma, which is attributed to crustal thickening, resulting from either subduction processes or terrane collision (Gaschnig, Vervoot, Lewis, and Tikoff, 2011; Lund, 1999).

Eocene intrusions related to the Challis Volcanic Field are common near the eastern margin of the Atlanta Lobe of the Batholith and include dikes, dike swarms, and stocks (Bennett and Knowles, 1985). The intrusions generally are porphyritic in texture and intermediate to felsic in composition. These younger Challis intrusions and associated volcanics, range in age from 51 Ma to 39 Ma, and were derived from both crustal and mantle sources. The Thunder Mountain Caldera Complex of the Challis Volcanic Field lies immediately east of the Project area and is described by Leonard and Marvin (1982) and Ekren (1985). It consists of predominantly felsic volcanic, pyroclastic, and epiclastic rocks that were erupted and deposited in subaerial and lacustrine environments.

Pleistocene-age valley glaciers created U-shaped valleys with over-steepened, talus-covered sides, and hanging valley tributaries with cirques and tarns in their upper reaches. U-shaped valleys also have lateral, terminal, and recessional moraines, remnants of moraine-dammed lakes, and glacial outwash deposits at their lower ends. Broadly glaciated areas have rounded hills with glacially scraped and scoured up-glacier slopes and ground-moraine covered down-glacier slopes. Modern Holocene-age stream drainage patterns indicate high rates of erosion and have deposited coarse-grained sedimentary fluvial deposits in floodplains often composed of a mixture of angular clasts from adjacent bedrock sources combined with more rounded reworked glacial deposits.

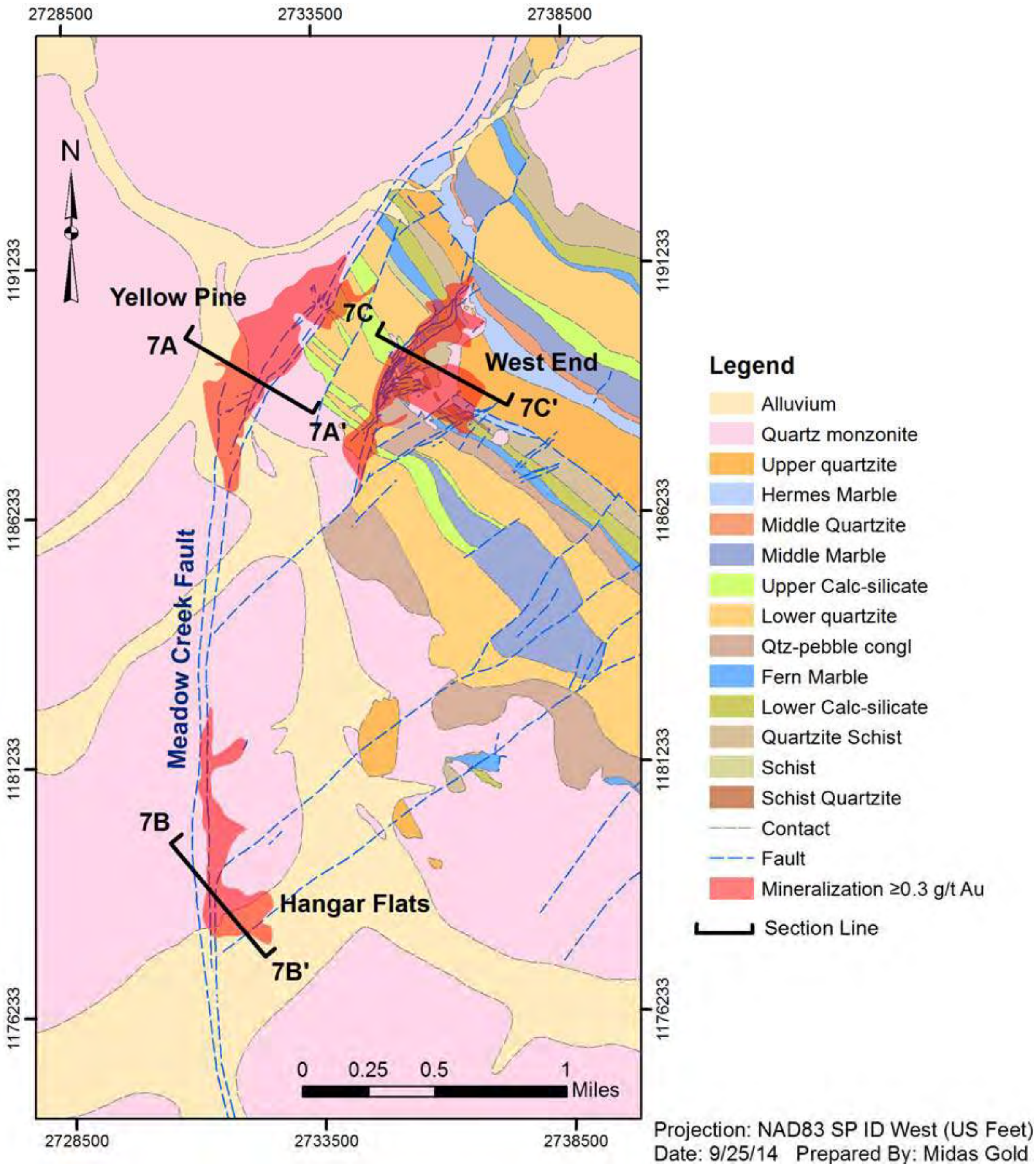
7.2 LOCAL GEOLOGY

7.2.1 Lithology

The Yellow Pine Deposit is hosted by intrusive phases associated with the Atlanta Lobe of the Idaho Batholith and by down-dropped blocks of metasedimentary rocks. The Hangar Flats Deposit is hosted by intrusive phases associated with the Atlanta Lobe of the Idaho Batholith. Other post-mineralization intrusive igneous rocks associated with the Challis Volcanics also occur within the Yellow Pine and Hangar Flats Deposits. The West End Deposit is hosted by metasedimentary rocks of the Stibnite roof pendant located within the Atlanta Lobe of the Idaho Batholith. Figure 7.2 illustrates the various lithologic units located within the Stibnite-Yellow Pine District (the District).

Numerous workers have described the stratigraphy and lithologic characteristics of the intrusive, metasedimentary, volcanic and unconsolidated rocks exposed in the Project area including: Larsen and Livingston (1920); Schrader and Ross (1926); Currier (1935); White (1940); Cooper (1951); and Smitherman (1985). The descriptions that follow are derived from these sources as well as from unpublished petrographic studies by past operators and Midas Gold.

Figure 7.2: Local Geology of the Stibnite Mining District



Quartz Monzonite

The dominant type of intrusive rock exposed in the District and intersected in drilling consists of Cretaceous, light to medium gray, equigranular, medium- to coarse-grained, granodiorite and quartz monzonite with distinctly peraluminous bulk rock geochemical compositional characteristics (Photograph 7.1). When unweathered and unaltered the quartz monzonite typically consists of approximately 25 - 30% quartz, 50 - 60% feldspar (mostly calcic oligoclase and the remainder microcline and orthoclase) and 5 - 10% biotite. Hornblende and other mafic minerals are rare. Accessory minerals include muscovite, chlorite, apatite, sphene and various carbonates and clay minerals. The unaltered quartz monzonite weathers to a white to light gray colored, chalky textured grus with rusty orange discoloration due to weathering and oxidation of biotite. Locally the biotites may show a weak alignment and the rock may be coarsely porphyritic with large feldspar phenocrysts. Zircon rims from an unaltered sample of biotite quartz monzonite from the Hangar Flats Deposit (drill hole MGI-10-21, 171-174 ft) were dated by U-Pb methods with LA-ICPMS to have an age of 91.2 Ma ± 2.2 Ma as reported by the IGS (Lewis, et al., 2014).

Photograph 7.1: Example of Quartz Monzonite in HQ Core

Alaskite

Alaskites occur as dikes, sills and segregations and range in width from less than 1 inch to over 30 ft. The alaskites are relatively siliceous, are typically fine-grained, sucrosic textured, and can be distinguished from the quartz monzonite by the lack of biotite or other mafic minerals (Photograph 7.2). The alaskite dikes can be coarsely crystalline to pegmatitic locally. Some alaskite dikes are unaltered and clearly crosscut altered quartz monzonite, and others are altered and cut unaltered quartz monzonite suggesting there may be several different ages of intrusions with similar mineralogy. The dikes may contain minor fine-grained disseminated euhedral magnetite and occasionally medium-grained euhedral arsenopyrite and often garnet. The alaskites typically occur as narrow 8- to 20-inch wide dikes in swarms that may range in overall width from a few feet to tens of feet across. Zircon rims from an altered and mineralized sample of alaskite from Hangar Flats Deposit drill core (MGI-10-20, 240 - 243 ft) were dated by the IGS and produced a U-Pb age of 87.9 Ma ± 4.9 Ma (Lewis, et al., 2014). Zircon tips from another drill core sample from the Yellow Pine Deposit (MGI-12-306 at 550 ft) were analyzed by ID-TIMS and produced a U-Pb age of 83.6 Ma ± 0.1 Ma (Gillerman, et al., 2014).

Hydrothermal alteration and the character of sulfide mineralization is the same as that described for the quartz monzonite. Sulfide mineralization occurs as veinlets, veins, stockworks, fissure filling, fault breccias, and massive sulfide veins or lenses. The biotite and magnetite are replaced by sulfides and/or pyrite as disseminations. Xenoliths of both unaltered and altered quartz monzonite have been observed within the alaskite dikes.

Photograph 7.2: Example of Alaskite in HQ Core

Pegmatite

Pegmatite dikes are coarsely crystalline consisting of large euhedral grains of interlocking potassium feldspar and quartz (Photograph 7.3). The pegmatite dikes range in width from 2 inches to more than 10 ft. Early pegmatite dikes cut through the quartz monzonite, but alaskite dikes have also been observed cutting through the early pegmatite dikes. Later pegmatite dikes cut through alaskite dikes.

Sulfide mineralization locally occurs as veinlets and up to several inches wide massive sulfide veins cutting through the pegmatite dikes.

Photograph 7.3: Example of Pegmatite in HQ Core

Biotite Granite

Biotite granite is exposed in several areas in the District and a large northeast-trending body is exposed and cut in drill holes between the West End Deposit and the Stibnite Pit, and has been informally named the Stibnite Stock. The biotite granite is typically fine- to medium-grained, equigranular with large black to dark brown biotite, and contains traces of hornblende, zircon and apatite as accessories (Photograph 7.4). Muscovite is present but in smaller quantities than biotite. The biotite granite crosscuts both the quartz monzonite and the metasedimentary sequence. Recent preliminary U-Pb LA-ICPMS isotopic dating by the IGS on zircons from outcrops of the Stibnite Stock in the Stibnite Pit produced an age of 84.9 Ma ± 2.0 Ma; a more precise age was recently reported from drill core from the Stibnite Stock in hole MGI-10-37 at 50ft., producing a concordant age of 85.7 ± 0.1 Ma with ID-TIMS methods (Gillerman, et al., 2014). Clasts of the biotite granite occur in mineralized breccias in the West End Deposit suggesting mineralization at least locally post-dates the stock and is consistent with recently reported $^{40}Ar/^{39}Ar$ isotopic dating of potassium feldspar selvages on quartz veins cutting the Stibnite Stock; the feldspar was dated at 50 Ma ± 0.4 Ma (Gillerman, et al., 2014).

Photograph 7.4: Example of the Biotite Granite in HQ Core with Local Fe Oxide

Granite

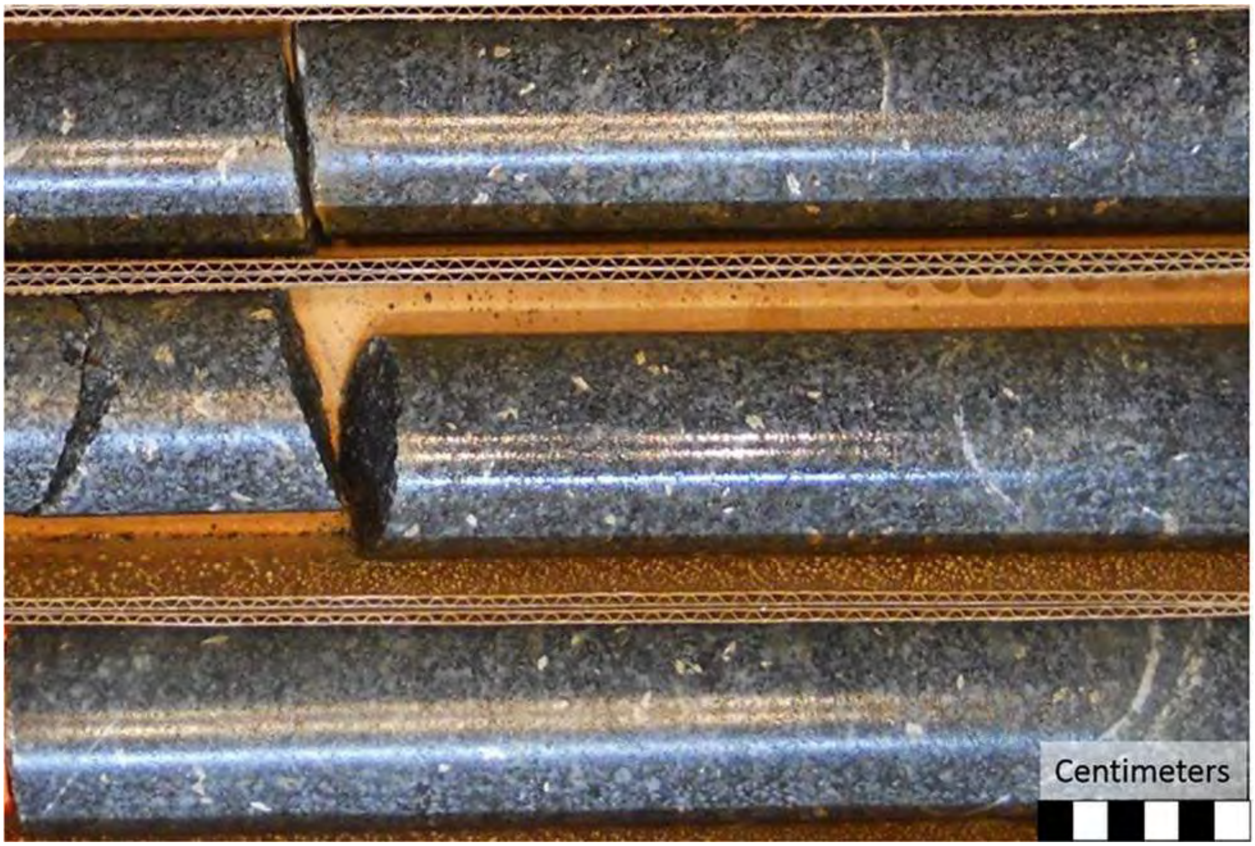
The granite is phaneritic, fine- to medium-grained, equigranular and typically light gray to white (Photograph 7.5). Principal components include feldspar, quartz, and fine-grained mica with accessories of magnetite, hematite, garnet, and sulfides (pyrite, arsenopyrite, and stibnite). Contacts with quartz monzonite are often gradational, which distinguishes the phyllosilicate-poor granites from the alaskites. No reliable isotopic dates have yet been determined for the granites. A large body of granite is exposed in the southwestern portion of the former Yellow Pine open pit and underlies the western portions of the Yellow Pine Deposit at depth. It likely represents a stock-like body based on three-dimensional interpretations of drill.

Photograph 7.5: Example of the Granite in HQ Core

Diorite

Diorite has been cut in several drill holes in the district at Yellow Pine, Hangar Flats, and near Scout and is exposed in the area around the Rabbit prospect. The diorites are fine- to medium-grained, and are often weakly magnetic due to the presence of magnetite and/or pyrrhotite (Photograph 7.6). Diorite clasts are observed as inclusions within quartz monzonite and occasionally crosscutting the quartz monzonite as well as the metasediments suggesting at least several different ages for the intrusions with dioritic composition. Primary mineralogy is plagioclase with equal parts amphibole and biotite (approximately 20% each) and very rarely quartz. Much of the amphibole may be an alteration product of pyroxene. Calcite or dolomite as well as magnetite occur as accessories. Trace amounts of sphene have also been observed within this lithology, likely as an alteration product. No isotopic dates have yet been determined for the diorites.

Photograph 7.6: Example of the Diorite in HQ Core



Rhyolite

Several rhyolite dikes are found within the district and are associated with the MCFZ. In the eastern side of the district they occur adjacent to the margin of the Thunder Mountain caldera. The rhyolites are aphyric to porphyritic, and are light to dark gray to beige in color when fresh, and weather to a distinctive green-brown mottled color due to weathering of magnetite and or sulfides forming iron-oxide stains (Photograph 7.7). The rhyolite contains sparse sub-inch sized, often resorbed quartz and feldspar phenocrysts within an aphanitic, often partially devitrified groundmass. Rhyolite dikes are up to 40 ft wide and are often sheared or strongly broken when they are located within fault zones. Xenoliths of mineralized quartz monzonite within the rhyolite have been observed in drill core and rhyolites likely were emplaced after the main pulses of mineralization. Both pyrite and stibnite have been observed in the rhyolites in small vugs and cavities suggesting remobilization of metals during emplacement. Based on similarities to dated rhyolites elsewhere in the area, these rhyolites are considered Tertiary in age.

Photograph 7.7: Example of the Rhyolite in HQ Core

Latite and Trachyte Porphyries

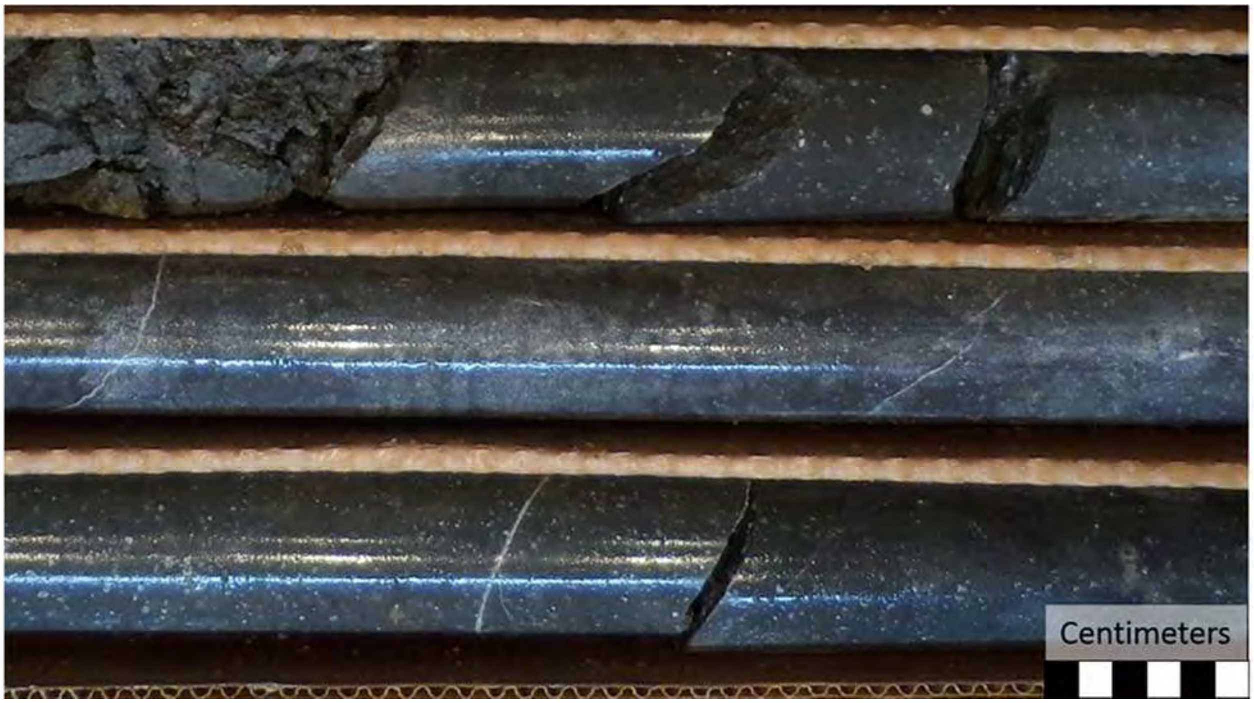
Porphyritic dikes of variable composition, but typically latite and trachyte, are common in faults throughout the district and occur as small plugs and sills in the eastern part of the Project area. The Pistol Creek Dike Swarm, located just southeast of the District and the Smith Creek Dike Swarm in Big Creek are both large regional-scale dike swarms of similar texture, mineralogy and composition and likely are of similar age. The dikes are light greenish gray in color when fresh and weather to an olive green to orange-gray color and often make a sticky, clay-rich soil likely due to alteration of devitrified glasses (Photograph 7.8). Phenocrysts of sanidine, andesine, biotite, and rare quartz are set in a groundmass of fine-grained feldspar ± fine-grained biotite. These dikes cross-cut the quartz monzonite and the granites and have been observed cutting the rhyolite dikes. A latite dike in drill hole MGI-13-383 sampled by the IGS at 285 ft from within the Yellow Pine Deposit produced an $^{40}Ar/^{39}Ar$ age of 45.9 ± 0.3 Ma (Gillerman, et al., 2014). This dike is well exposed in the Yellow Pine Deposit and, although moderately altered, appears to be later than the main pulses of mineralization at Yellow Pine. Fragments of a similar lithology occur as clasts in mineralized breccias within the West End Deposit.

Photograph 7.8: Example of Latite in NQ Core

Diabase

Diabase dikes, up to 50 ft wide, often occur within or adjacent to fault zones within the district. Historic literature occasionally noted these as lamprophyres. Based on crosscutting relationships, the dikes are likely Eocene or younger. They typically are brecciated and heavily fractured when they occur within structures. They are typically aphanitic to very finely porphyritic in texture, medium to dark green in color when fresh, containing small partially resorbed grains of pyroxene and hornblende, with phenocrysts making up less than 5% of the rock unit within an aphanitic groundmass primarily of plagioclase feldspar (Photograph 7.9). Magnetite is a common accessory and is generally magnetic. Locally, they contain circular to ovoid, calcite-filled amygdules similar in appearance to outcropping Eocene basalt flows associated with the latest stages of Eocene volcanism within the adjacent Thunder Mountain Caldera and to the west in younger Miocene basalt flows in Long Valley. Rarely, xenoliths of rhyolite dike material have been found as fragments within the diabase dikes, indicating that diabase dikes are the youngest rock unit and were emplaced after the main phases of mineralization. However stibnite has been observed in the diabases in small vugs and cavities along late fractures suggesting remobilization of metals during emplacement.

Photograph 7.9: Example of Diabase in NQ Core

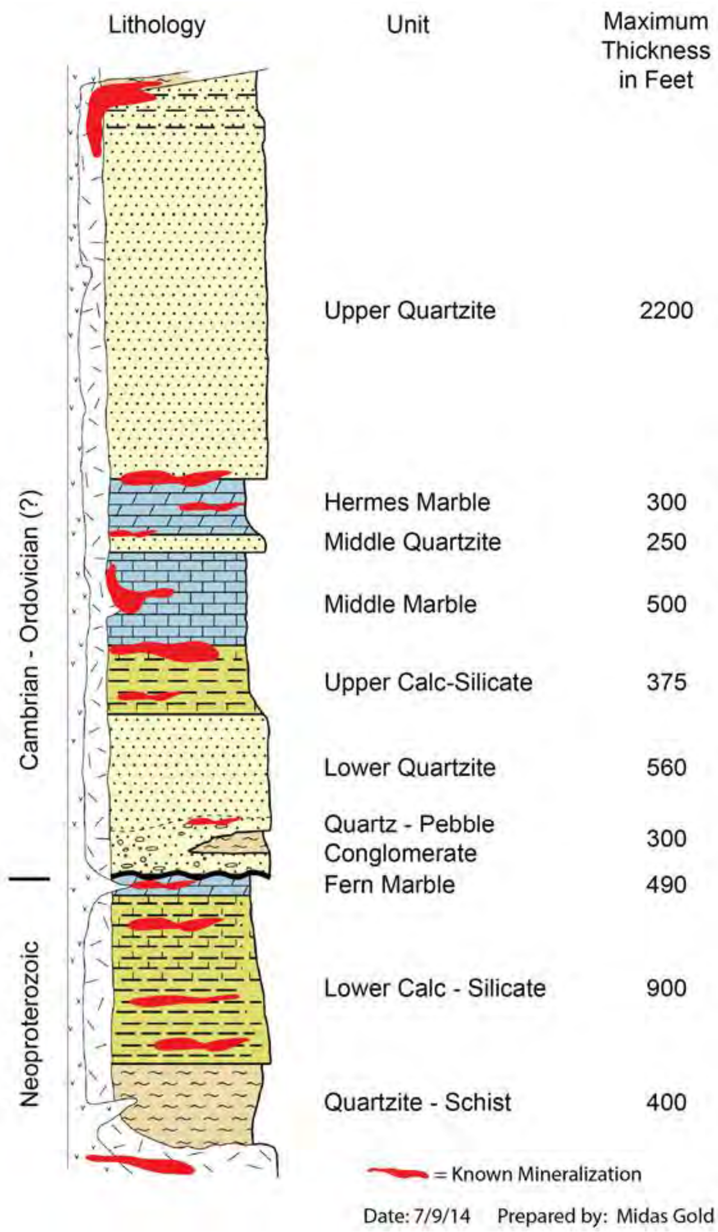


Metasedimentary Rocks

Early workers believed that the rocks of the roof pendant were Proterozoic in age, partly because of their proximity to the Belt sedimentary basin, however, recent work has determined they at least some of the rocks are likely Paleozoic in age. Based on coral and bryozoan fossils, researchers in the early 1980s used biostratigraphy to place the Stibnite metasedimentary package in the Ordovician Period (Lewis and Lewis, 1982). Additional bryozoan fossils were discovered in 2012 by the IGS from the Hermes Marble near Sugar Creek. Detrital zircons recovered by the IGS from within the suite show ages in the Mezo- and Neo-Proterozoic (Lewis, et al., 2014).

Early rudimentary stratigraphy was presented by Currier (1935), but Smitherman (1985) constructed a more detailed and comprehensive stratigraphic column of the Stibnite roof pendant (Figure 7.3). The metasedimentary rock units are divided into ten informal units. They are, in ascending stratigraphic order: Quartzite-schist, Lower Calc-silicate, Fern Marble, Quartz Pebble Conglomerate, Lower Quartzite, Upper Calc-silicate, Middle Marble, Middle Quartzite, Hermes Marble, and Upper Quartzite. The following descriptions are based mainly on Smitherman's work (1985) and include additional information from various unpublished studies completed by previous operators and by Midas Gold.

Figure 7.3: Stibnite Roof Pendant Stratigraphy



Source: modified from Smitherman, 1985

Quartzite-Schist

This unit is up to 460 ft thick and is apparently the oldest unit exposed in the immediate Project area. Exposures are confined to two northwest trending belts: one along the northeast roof pendant border and one extending through the center of the roof pendant. The lower contact of the northeast belt is with the Idaho Batholith and a major fault forms the lower contact of the central belt. Schistosity is moderately developed with 4-inch- to 4-foot-thick interbeds of quartzite and schist forming distinct compositional banding likely reflecting original lithologic bedding (Photograph 7.10). Intermediate lithologies between quartzite and schist are common and the unit is subdivided into quartz-mica schist, garnet-bearing quartz-biotite schist, and micaceous quartzite. The aluminous quartz-mica schist consists of quartz-muscovite-biotite ± andalusite + sillimanite + chlorite. The quartz biotite schist is 80% fine-grained quartz, 10% biotite grains (in biotite rich layers) which define a foliation, and 2% to 3% almandine garnet porphyroblasts. The micaceous quartzite contains over 90% quartz and 5% to 10% muscovite, which has developed a weak schistosity. Traces of biotite, sphene, zircon, tourmaline, and opaque minerals occur as accessories. Based on regional mapping in the Big Creek area and northeast of the District by the IGS, this unit is interpreted to be Neoproterozoic in age (Lewis, et al., 2014).

Photograph 7.10: Example of the Quartzite Schist in HQ Core

Lower Calc-Silicate

This unit is 165 ft to 900 ft thick, consisting of thin-bedded siltites and calc-silicate bearing rocks. The contact between the quartzite-schist and the calc-silicate sequence appears to be gradational. Minor folds are common and probably account for much of the variation in thickness. The unit contains grey quartz-feldspathic layers with alternating green calc-silicate beds in the lower portion and light grey calcitic marble with green calc-silicate interlayers in the upper portion (Photograph 7.11). The dark layers are composed of fine oligoclase, microcline, and quartz. Xenoblastic epidote constitutes 20% to 90% of the calc-silicate layers, with minor hornblende, actinolite, and scapolite. The calcareous calc-silicate rock contains interlayers of calcite marble and calc-silicate rocks. Calc-silicate minerals include xenoblastic diopside, pale green tremolite and actinolite, and minor scapolite. Epidote occurs as very fine grains between the calcite and quartz rich layers and as coarse grains with tremolite and actinolite intergrown with pyrite. Accessory minerals include phlogopite, rare sphene, and allanite. Locally the rocks have been altered to a coarse-grained skarn assemblage of garnet-epidote-diopside, calcite, pyrite, and iron oxide.

Photograph 7.11: Example of the Lower Calc-Silicate in HQ Core

Fern Marble

The Fern marble overlies the lower calc-silicate and reaches a maximum thickness of about 500 ft. Fresh marble is light gray to blue gray and weathers to a light yellow to white leaving sucrosic-textured outcrops and poorly developed sandy soils (Photograph 7.12). The marble consists of coarse dolomite grains, rare quartz grains, and traces of brown amorphous material that may reflect the former presence of carbon residues. Green gray calc-silicate marble is locally common within 500 ft of the batholith contact. One specimen from the West End Pit is composed of 60% green diopside, 40% colorless to green tremolite/ actinolite, and rare phlogopite, forsterite, and dolomite or calcite.

Photograph 7.12: Example of the Fern Marble in HQ Core with local Fe Oxide

Quartz-Pebble Conglomerate

The quartz-pebble conglomerate is a coarse-grained, pebbly quartzite unit, which contains lenses of pebble conglomerates and bodies of quartz-mica schist (Photograph 7.13). The contact with the Fern marble is well exposed and likely represents an unconformity. The quartz pebbles are coarse, irregular to polygonal grains with flattened quartz grains and muscovite as the matrix. Small schist lenses occur locally and consist of quartz-muscovite, biotite, sillimanite, and andalusite. The unit is thickest and best exposed in the area near the Fern Mine and thins and appears to pinch out towards the West End Deposit. The unit clearly crosscuts the Fern marble along an unconformity surface well exposed in the Fern Mine area on the east side of the District. Detrital zircon dated using Laser Ablation Inductively Coupled Plasma Mass Spectrometry (LA-ICP-MS) methods and regional relationships suggest this unit is likely Neoproterozoic in age and possibly correlative in age to the Neoproterozoic Caddy Canyon quartzite exposed near Pocatello in southeast Idaho (Lewis, et al., 2014).

Photograph 7.13: Example of the Quartz-Pebble Conglomerate in HQ Core with Fe Oxide

Lower Quartzite

The quartz-pebble conglomerate unit grades upward into a muscovite-bearing quartzite that is 295 ft to 560 ft thick. The quartzite is typically light gray and commonly shows dark gray streaks, which appear to be relict bedding (Photograph 7.14). Outcrops are large and bold, occurring along ridges and on slopes. The rock weathers into large blocks and vast talus fields. Thin sections show that the quartzite is 95% fine to very coarse-grained quartz. Muscovite grains make up to 5% to 10%, quartz is up to 85%, and andalusite may be 2%.

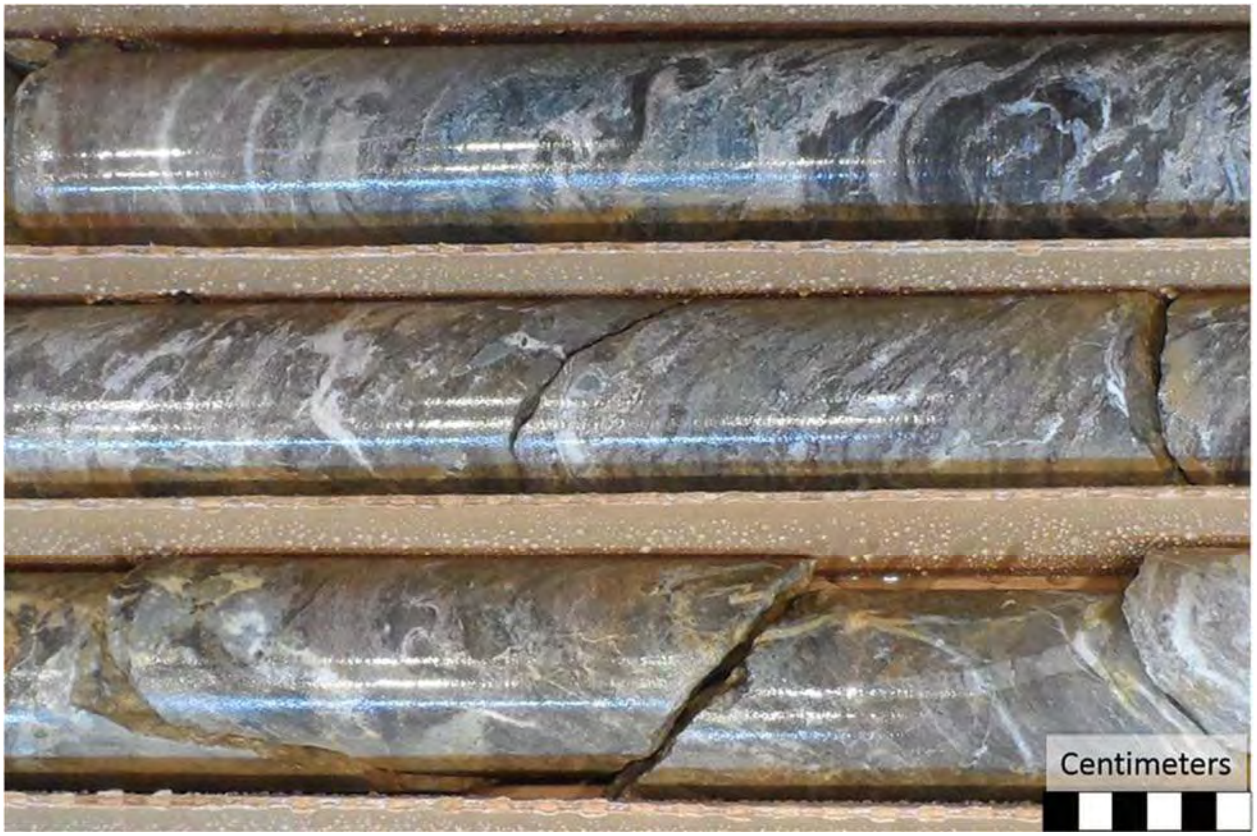
Photograph 7.14: Example of the Lower Quartzite in HQ Core

Upper Calc-Silicate

The upper calc-silicate consists of biotite, plagioclase, calc-silicate rock (Photograph 7.15). The unit thickness varies from about 100 ft to about 375 ft, likely due to zones of isoclinal folding. The internal stratigraphy of the unit includes four sub-units.

The lower plagioclase calc-silicate rock is dense, laminated, dark gray, and weathers to gray or red-brown. Thin sections show 70% plagioclase, 10% diopside, 10% tremolite/actinolite, and 5% fine-grained biotite. The middle plagioclase-biotite rock is similar to the lower unit with the addition of plagioclase-biotite layers. The upper unit is a massive calcareous, plagioclase calc-silicate rock with 35% labradorite, 30% scapolite, 30% diopside, and minor calcite. The uppermost unit is a laminated calc-silicate and calcitic marble rock. This unit varies up to 195 ft in thickness on the northern limb of the syncline. The calc-silicate layers form thin (0.4 inch) ribs above the easily weathered marble layers. Minor interbedding folds are common. The calc-silicate layers are about 50% scapolite and 50% fine diopside grains. The marble layers are approximately 0.6 inch thick and contain over 95% calcite with minor scapolite and diopside.

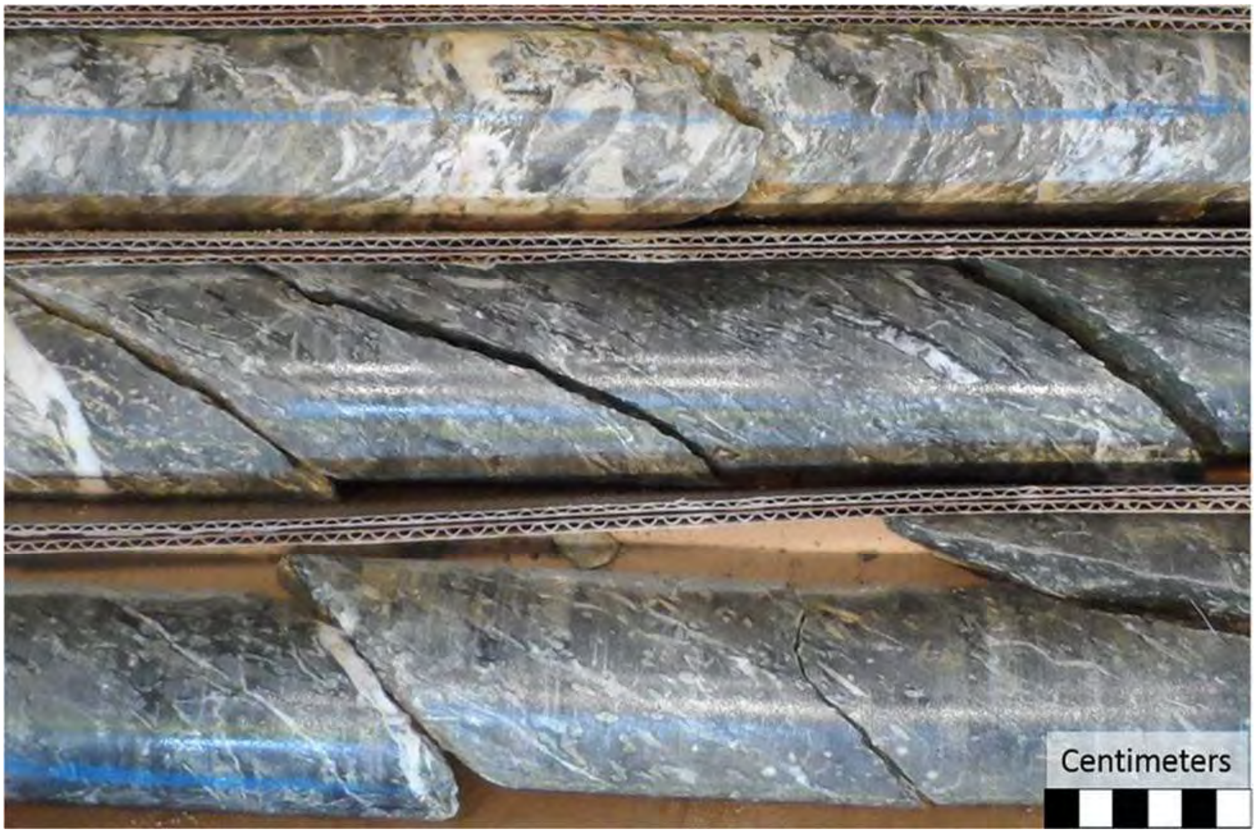
Photograph 7.15: Example of the Upper Calc-Silicate in HQ Core



Middle Marble

The upper calc-silicate unit grades upward into a calcitic marble unit that is 260 ft to 490 ft thick. The unit is dominantly a massive, blocky, thick bedded blue-gray finely crystalline limestone interbedded with thinner light gray thin-bedded (1 inch) laminated marble (Photograph 7.16). The rock is 80% to 99% calcite with minor biotite, diopside, and graphite.

Photograph 7.16: Example of the Middle Marble in HQ Core



Middle Quartzite

A 30 ft- to 250 ft-thick quartzite unit lies above the Middle Marble. It is a light gray, fine- to coarse-grained, vitreous quartzite. Accessory minerals are K-feldspar, sericite, graphite, leucoxene, zircon, and iron oxide. The rock is locally very porous due to hydrothermal leaching, and sparse alunite grains probably formed during hydrothermal activity. Carbonate cement is locally present, as well as rare biotite schist bodies near the lower contact. Stratigraphic relationships are important for identifying this unit, as the texture can often be similar to the Lower Quartzite, which can be seen in Photograph 7.14.

Hermes Marble

The Middle Quartzite is overlain by 195 ft to 295 ft of dolomite marble. The lower 195 ft consist of a light gray massive dolomite marble (Photograph 7.17). This contains 80% dolomite and 20% altered tremolite porphyroblasts. Alteration of the tremolite is probably hydrothermal and resulted in clay replacing 90% of the tremolite. Minor pyrite and iron oxide are locally present. The upper portion is a gray, laminated marble that has essentially the same mineralogy, but is generally unaltered. Throughout its outcrop area and in underground workings and drill holes within the Cinnabar Mine complex east of the District, the Hermes is often silicified and converted to maroon to grey-red jasperoids.

Photograph 7.17: Example of the Hermes Marble in HQ Core

Upper Quartzite

Overlying the Hermes Marble is a quartzite unit with minor siltite. Thickness varies from 1400 ft to 2200 ft. The unit forms large, bold outcrops of cliffs and ridges. In thin section the quartzite is nearly pure quartz with less than 3% muscovite. Locally, black quartzite contains intergranular graphite. Accessory minerals include zircon, magnetite, sericite, and secondary iron oxide, after pyrite. Laminated gray siltite occurs in the upper portion of the unit. The siltite is composed of 70% to 90% fine quartz grains, with the remaining 10% to 30% biotite and minor muscovite. Stratigraphic relationships are important for identifying this unit, as the texture can often be similar to the Lower Quartzite, which can be seen in Photograph 7.14. Preliminary detrital zircon dating as reported by the IGS suggests the unit is likely an age equivalent with the Ordovician Kinnikinic Quartzite of the Bayhorse area along strike to the southeast in southeast Idaho (Lewis, et al., 2014).

7.2.2 Structure

Regional- and district-scale structural trends are broadly parallel to the trace of the relict rifted western edge of the continent, suggesting it was a fundamental control on the geometry of the miogeocline, subsequent contractional orogenic events and development of the suture zone. Lund, et al. (2003) suggested that the rifted margin contained two segments, interpreting the variability between pendant stratigraphy as reflecting the effects of northwest-striking asymmetric extensional segments divided by northeast-striking transform and transfer segments. These earlier large scale crustal features controlled the provenance and spatial distribution of sedimentary lithologies and also likely played a role in where subsequent intrusive and volcanic activity developed with pre-existing zones of weakness providing conduits for ascending magmas and circulation of hydrothermal fluids (Georgis, Tikoff, Kelso and Markley, 2004).

Several major regional scale structural features cut through the Project area along with smaller subsidiary structures. Historic surface and underground mining records, field mapping, data from oriented drill core and geophysical surveys indicate three dominant trends within the district that are similar to those found in more well studied areas to the west in the Long Valley area. Structural elements show a wide variety of characteristics including thrust, low angle normal, high angle reverse and normal, and strike slip movement.

Large, north-south striking, steeply dipping to vertical structures occur in the central and eastern portions of the property and include: the MCFZ; the Scout Valley Fault Zone (SVFZ); and the Mule Fault Zone (MFZ). These features exhibit pronounced gouge and multiple stages of brecciation, suggesting multiple periods of movement. They are poorly exposed and are recessive weathering and often are found under or along the flanks of glacially carved valleys. Interpretation of kinematic indicators in underground and surface exposures and oriented drill core suggest these faults had early high angle reverse movement followed by right lateral displacement, but due to structural complexity variations in sense and amount of relative displacement are common. These north-south faults are often associated with east-west and northeast-southwest trending splays and dilatant structures and locally appear to truncate the northeast-trending features, but due to lack of exposure the relationships are unclear.

Large, northwest-southeast trending geophysical features occur cutting through and within the metasedimentary rocks of the roof pendant and continue to the northwest across batholith rocks and across through the younger caldera sequence to the southeast suggesting these features have at least some movement after development of the north-south and northeast elements. A distinct “break” in rather continuous mineralization in the main Yellow Pine deposit area and the Homestake area may be related to one of these northwest structural features.

The MCFZ is the dominant structure associated with the Hangar Flats Deposit. A jog in the fault occurs adjacent to the main deposit and kinematic indicators (from historic operators with access to the underground workings and from Midas Gold geotechnical drilling and oriented core studies) show an early reverse sense of movement followed by right lateral strike slip displacement. This jog likely created a dilatant zone allowing hydrothermal fluids to pervasively alter and mineralize the area near the bend. A pronounced and pervasive set of northeast to east-northeast striking, shallow northwest dipping joints and an alaskite dike swarm occurring adjacent to the MCFZ are likely reflecting the presence of dilatant splays generated during movement along the MCFZ.

Mineralization in the Yellow Pine Deposit is also structurally controlled and localized by the MCFZ, a generally north to northeast striking, steeply west-northwest dipping, complex fault zone; and north striking gently west dipping conjugate splay or cross structures associated with the MCFZ. The main body of mineralization in the Yellow Pine pit area is associated with a dilatant bend in the MCFZ, where its strike changes from a linear north-south trend to a more north-easterly trend. Early reverse movement and later right lateral strike slip movement along this fault created a large area of fracturing and open space allowing hydrothermal fluids to pervasively alter and mineralize the rocks within the area of the bend. Historic operators mapped several large faults here and they are discussed in the mineral resources section of this Report.

The West End Fault Zone (WEFZ) is the predominant structure associated with the West End Deposit. The main fault zone consists of three high angle faults, all striking along an azimuth of approximately 030° and dipping 50° to 75° to the southeast. The width of the fault zone as measured between the footwall and the hanging wall faults varies from 100 ft to 295 ft. Several subsidiary structures exist on the northern and southern ends of the deposit both west and east of the primary WEFZ, but are poorly defined at present and are not well exposed. Several east-northeast striking structures appear to splay off the primary structural zone and include the Splay Fault, Stibnite Fault, and Northeast Extension Fault structures. The subsidiary structures have strikes ranging from azimuth 060° to 090° and dip steeply north and south. Based on the relative offsets of the metasediments, and kinematic indicators, the WEFZ has experienced right lateral and probably normal (down to east) offset.

7.2.3 Alteration

Intrusive Rocks

Mineralization in intrusive rocks from the Yellow Pine and Hangar Flats deposit were described by White (1940) and Lewis (1984). Lewis subdivided alteration of granitic rocks as an early sodium metasomatism followed by potassium metasomatism and this was subsequently followed by multistage potassium alteration, which he described as hydrothermal in origin. Lewis's work outlined four phases of hydrothermal potassium alteration following the metasomatism. Alteration is not typically texture destructive at hand specimen scale and many of the primary textural relations of the typical quartz monzonite host rock still appear to be evident. However, microscopic examination indicates this early alteration is highly pervasive.

The earliest gold bearing alteration phase is typified by hydrothermal replacement of plagioclase and microcline by adularia, quartz flooding, and alteration of biotite to sericite. The adularia replacement of plagioclase is pervasive and is associated with distinct geochemical changes, including increased potassium content associated with sodium depletion, and often calcium and magnesium depletion as well, presumably due to the destruction of sodium-bearing feldspar phases and replacement with potassium-rich feldspars and replacement of biotite with sericite. This phase is associated with the introduction of very fine-grained, disseminated, euhedral hydrothermal pyrite. Arsenopyrite, pyrrhotite, or other sulfides are rare. This early pyrite is disseminated throughout the matrix of the quartz monzonite, but is concentrated in biotite and to a lesser extent in the feldspar phases.

The second gold bearing alteration phase also includes adularia replacing plagioclase and microcline, addition of quartz, and sericitization of biotite, but also includes introduction of pyrite, arsenopyrite, and minor pyrrhotite. By far, pyrite is the dominant sulfide phase, making up over 90% of the sulfides, and the sulfides exhibit distinctive microcrystalline textures. The sulfides occur as elongated "blebs" of sulfides in the sericite which replaces the biotite. These sulfide "blebs" may be randomly oriented or more typically are oriented parallel to the original cleavage in the micas. The arsenopyrite is typically represented by euhedral grains surrounded by pyrite. In small irregularly shaped patches, minor amounts of pyrrhotite may occasionally be present. Characterization studies performed during metallurgical work indicate the earlier sulfides are likely higher temperature, have more arsenic and more gold and that later sulfides often have developed at the expense of the earlier sulfides and document various morphological changes over time (Martin and Palko, 2011a; Martin and Palko, 2011b; Palko, 2012). Carbonates (dolomite and calcite) were introduced and usually occur as partial replacements of adularia or plagioclase or in the sericitized groundmass.

The third phase of potassium alteration represents a period when sulfides were not precipitated in significant quantities and the majority of the alteration occurs as the coarse-grained sericite replacement of adularia.

The fourth alteration event is distinguished by open space filling and is represented by dolomite, calcite, quartz, and sericite precipitating in small cavities and along fractures and as fissure-filling veinlets with pyrite and arsenopyrite. Coarse-grained stibnite veins are commonly associated with this stage.

Metasedimentary Rocks

Secondary silica, as veins and disseminations, is the most pervasive alteration of the mineralized material. Silica has replaced and permeated the rock within and near the major fault zones; in places silica is over 90% of the rock mass. Quartz occurs in stockwork veins and veinlets, as disseminations, in coliform bands, and with reticulate textures. Vugs formed by leaching of feldspar grains or formed in extensional fractures are either partially filled with euhedral quartz + scheelite + gold and commonly with manganese and iron oxides, or they are completely filled with secondary quartz, forming a quartz-eye pattern in the rock (Cookro, 1989). As noted earlier, higher temperature quartz veins are cut by veins with distinctive lower temperature assemblages and fluid inclusions (Cookro, et al., 1987).

Plagioclase is commonly sericitized, although not necessarily in the mineralized zones. Feldspar grains are commonly argillized in the fault zones. Clay minerals that formed from the alteration of feldspar grains are commonly leached out, and the resulting void is filled with euhedral clear quartz. Altered silicates were an important source of secondary silica.

Calcite, dolomite and locally ankerite and siderite, occur as discrete grains and as coarsely crystalline carbonate in veins and cementing breccia fragments. Carbonates are sometimes dark in color and contain manganese and iron oxides and abundant opaque minerals, which include fine grains of sulfides (Cookro, et al., 1987) indicating they are likely hydrothermal in origin.

In the West End Deposit, gold concentrations occur within fractured, metasedimentary rocks. Although calc-silicates are the most favorable host rocks, all lithologies host mineralization. Within the oxidized zone, gold is tied up in iron oxides and hydroxide oxidation products of primary sulfides. Silica and potassium feldspar flooding and veining are manifestations of alteration associated with gold mineralization.

7.2.4 Mineralization

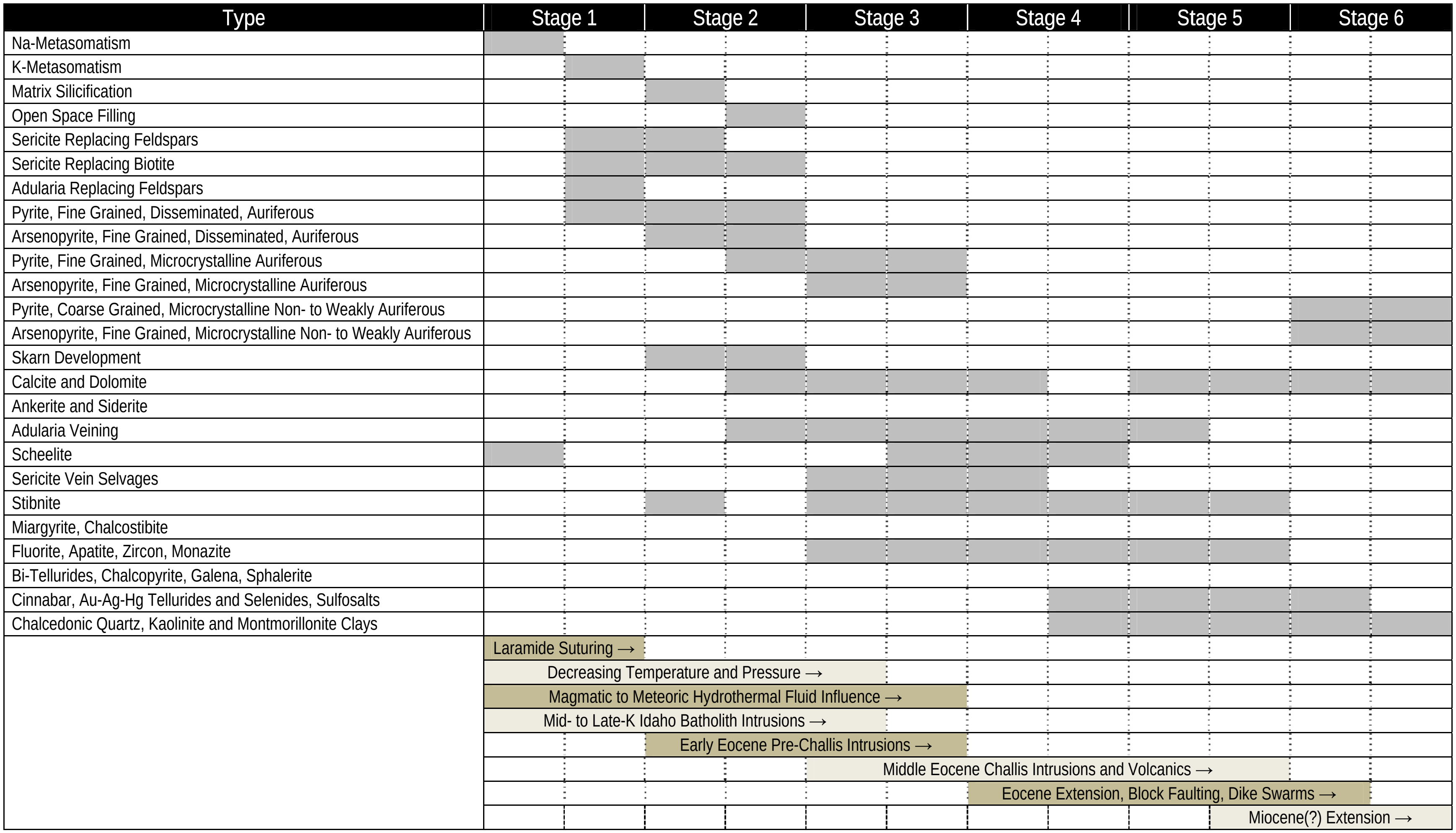
Intrusive hosted precious metals mineralization typically occurs in structurally prepared zones in association with very fine-grained disseminated arsenical pyrite (FeS_2) and to a lesser extent arsenopyrite (FeAsS). Base metal sulfides are uncommon. Mineralogical studies of sulfide morphology and mineral chemistry were completed for metallurgical process flow sheet testing using x-ray diffraction (XRD), dynamic secondary ion mass spectrometry (SIMS), QEMSCAN®, mineral liberation analyzer (MLA), and petrographic studies (Martin and Palko, 2011a; Martin and Palko, 2011b; Martin and Palko, 2011c). These studies, combined with past academic research (White, 1940; Cooper, 1951; Lewis, 1984; Cookro, et al., 1987) indicate that there are multiple periods of pyrite development and associated precious metals mineralization. Arsenical pyrite is the primary host for gold mineralization, and gold only rarely occurs as discrete particles and, if so, typically only in rare sub-micron size particles, but the vast majority of the gold instead occurs in solid solution within the pyrite crystal lattice. Arsenopyrite is the only other significant gold-bearing sulfide mineral in the intrusive hosted deposits. Base metals (except for arsenic, antimony, and tungsten) are rare and occur at very low concentrations, at or below typical crustal abundance levels. Various oxidized products of the weathering of the primary sulfides are found in the intrusives, including goethite, hematite, jarosite, and scorodite and host precious metal mineralization in the oxidized portions of the deposits.

Antimony mineralization occurs primarily associated with the mineral stibnite (Sb_2S_3). Other antimony-bearing phases include miargyrite ($AgSbS_2$), gudmundite (FeSbS), chalcostibite ($CuSbS_2$), tetrahedrite [$(Cu, Fe)_{12}Sb_4S_{13}$], and owyheeite [$(Pb)_{10}(Ag)_{3\text{-}8}(Sb)_{11\text{-}16}(S)_{28}$]. There is a weak, but persistent association of volumetrically small, typically <0.25%, base metal mineralization associated with the antimony mineralization and includes rare occurrences of chalcopyrite ($CuFeS_2$), galena (PbS), sphalerite (ZnS) and molybdenite (MoS_2). Zones of high grade, silver-rich mineralization locally occur with antimony and are related to the presence of pyrargyrite (Ag_3SbS_3), hessite (Ag_2Te) and acanthite (Ag_2S).

Tungsten mineralization is typically and essentially exclusively associated with the mineral scheelite ($CaWO_4$). Observations suggest tungsten occurs late in the paragenesis, but precedes the stibnite mineralization since stibnite has been found in numerous past studies cementing veins and brecciated scheelite fragments.

Although mercury mineralization is rare in the area of the three main deposits and in the west side of the district, studies of the mineral occurrences to the east in the Cinnabar district, where mercury was historically produced, indicate the primary mercury-bearing minerals are cinnabar (HgS) and coloradoite (HgTe) and to a lesser extent tiemannite (HgSe) and amalgam (HgAg).

Figure 7.4: Stibnite – Yellow Pine District Paragenesis



Source: modified from Lewis, 1984; Cookro, et al., 1988; Blue Coast Metallurgy, 2012

Metasediment-hosted mineralization has a similar sulfide suite and geochemistry, but with higher carbonate content in the gangue and a much more diverse suite of late stage minerals. As in the intrusive-hosted mineralization, gold is associated with very fine-grained arsenical pyrite and is tied up in the pyrite lattice. Rarely, submicron sized native gold occurs as inclusions and along fractures, and may be disseminated in highly fractured zones and may produce locally high grades and a minor nugget effect. Metallurgical test work completed by Midas Gold to date suggests around 20% of the gold in the West End metasediment-hosted mineralization may be particulate in nature, but extremely fine-grained.

The paragenesis of mineralization in the district and immediate area has been described by various workers including: Currier (1935); White (1940); Leonard and Marvin (1982); Lewis (1984); Cookro, et al. (1987). Figure 7.4 graphically outlines the primary stages of alteration and mineralization as currently interpreted. Additional studies by Midas Gold contractors and consultants and academic researchers from the IGS and USGS are ongoing.

7.3 MINERALIZED ZONES

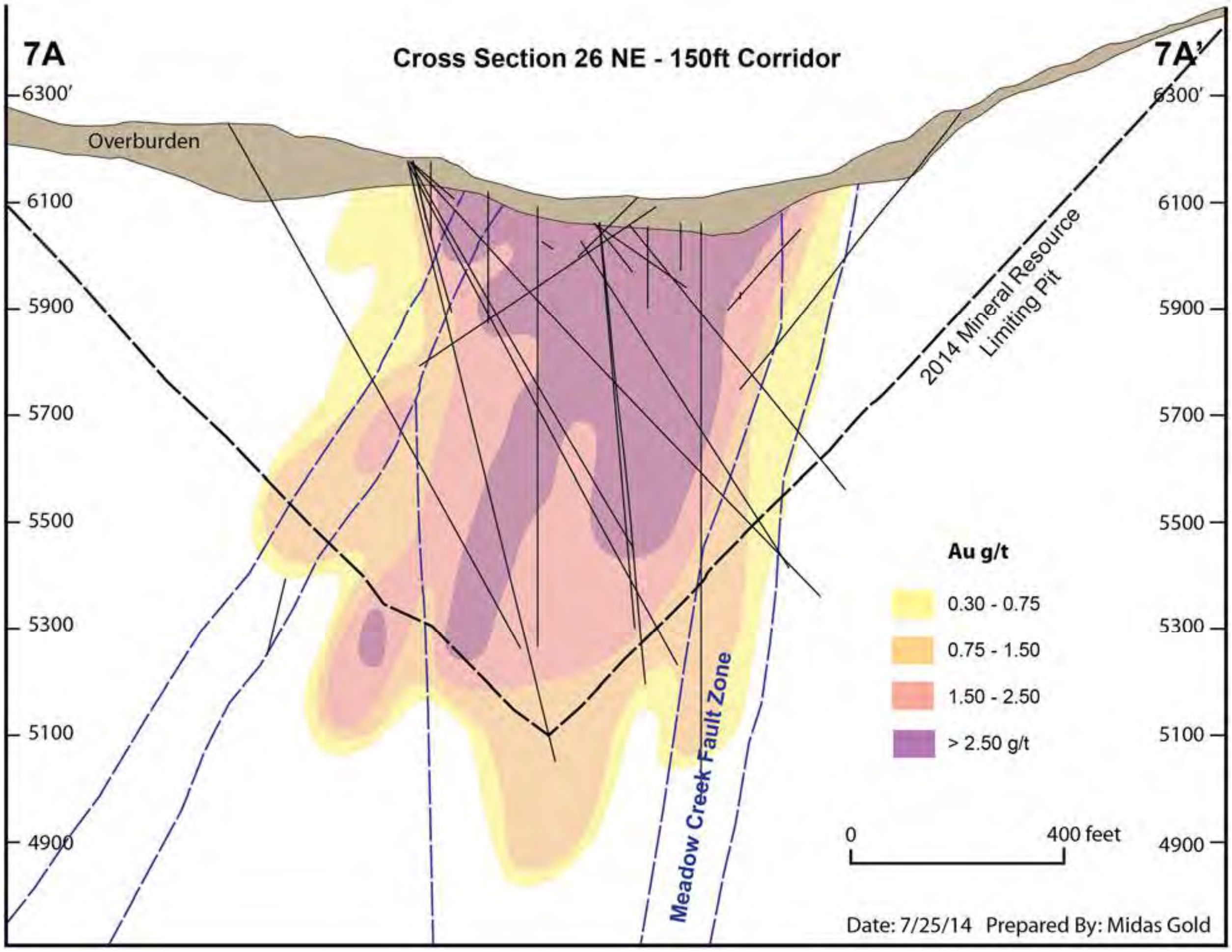
7.3.1 Yellow Pine Deposit

Mineralization of the Yellow Pine Deposit is structurally controlled and localized by the MCFZ and related structures. Mineralization styles, intensity of mineralization, widths and intensity of alteration vary relative to distance from the bend in strike of the MCFZ. Variography and stereonet plots of observed outcropping structures, mineralized features and data from modeling of oriented and un-oriented drill core, along with compilation of historic open pit and underground geologic information, has defined a series of domains outlining areas with common characteristics. Gold and antimony have different geochemical signatures, geometries, and locally used different structures during deposition. Structures and fractures open to circulating hydrothermal fluids during gold deposition were not necessarily open for antimony deposition. The deposit shows some apparent zonation with gold occurring throughout the deposit footprint, but with antimony and tungsten primarily in the central and southern portions of the deposit.

The dominant fault directions mapped underground and in the open pits by Bradley Mining Company geologists in the 1938-1952, by White in 1940-41, by Cooper 1950-1951, and also observed along the former open pit benches by Midas Gold geologists in 2012, trend north-south, northeast, and east-northeast. However, the controls for antimony mineralization show more northwesterly trends. The different geometries of antimony and gold distribution suggest different controls for mineralization – antimony is more strongly influenced by northwest fracturing and gold is more strongly influenced by northeast and east-northeast structures. White (1940) interpreted all strike-slip faulting as post-mineral whereas Cooper (1951) suggested there was significant post-mineralization movement between periods of early gold mineralization and later antimony-tungsten mineralization. Midas Gold's current interpretations on the relative timing of gold versus antimony mineralization are similar to those interpreted by Cooper (1951).

Mineralization at the south end of the Yellow Pine Deposit exhibits strong, steeply west- and east-dipping north-south oriented structural controls and occurs in a narrow 80-ft- to 165-ft-wide corridor along the footwall (east side) of the MCFZ. In the central domain of the deposit, numerous structural elements intersect and mineralization occur along east to east-northeast striking and west to west-northwest striking, north-dipping dilatant structures within a larger northeast-trending structural corridor. Both northeast and north-south striking structural elements may control mineralization along with the dilatant structures which occur at relatively high angles to the main shear zone. Mineralization in this area appears pipe-like in cross sections, but in long sections exhibits pronounced northeast and northwest plunges reflecting the interplay of the primary northeast structures and secondary splays and dilatant features which occur at high angles to the main MCFZ. The multiple structural features provided significant pathways to mineralizing hydrothermal solutions and the mineralization here is the highest grades and ranges from 165 ft to over 650 ft in true thickness and can be traced down dip for over 1,300 ft (Figure 7.5).

Figure 7.5: Yellow Pine Mineralized Zone



Note:
The Yellow Pine mineralized zone was mined by underground methods between approximately 1939 and 1948. For simplicity, the underground workings are not shown on the cross section but are quantified in Section 14 of this Report.

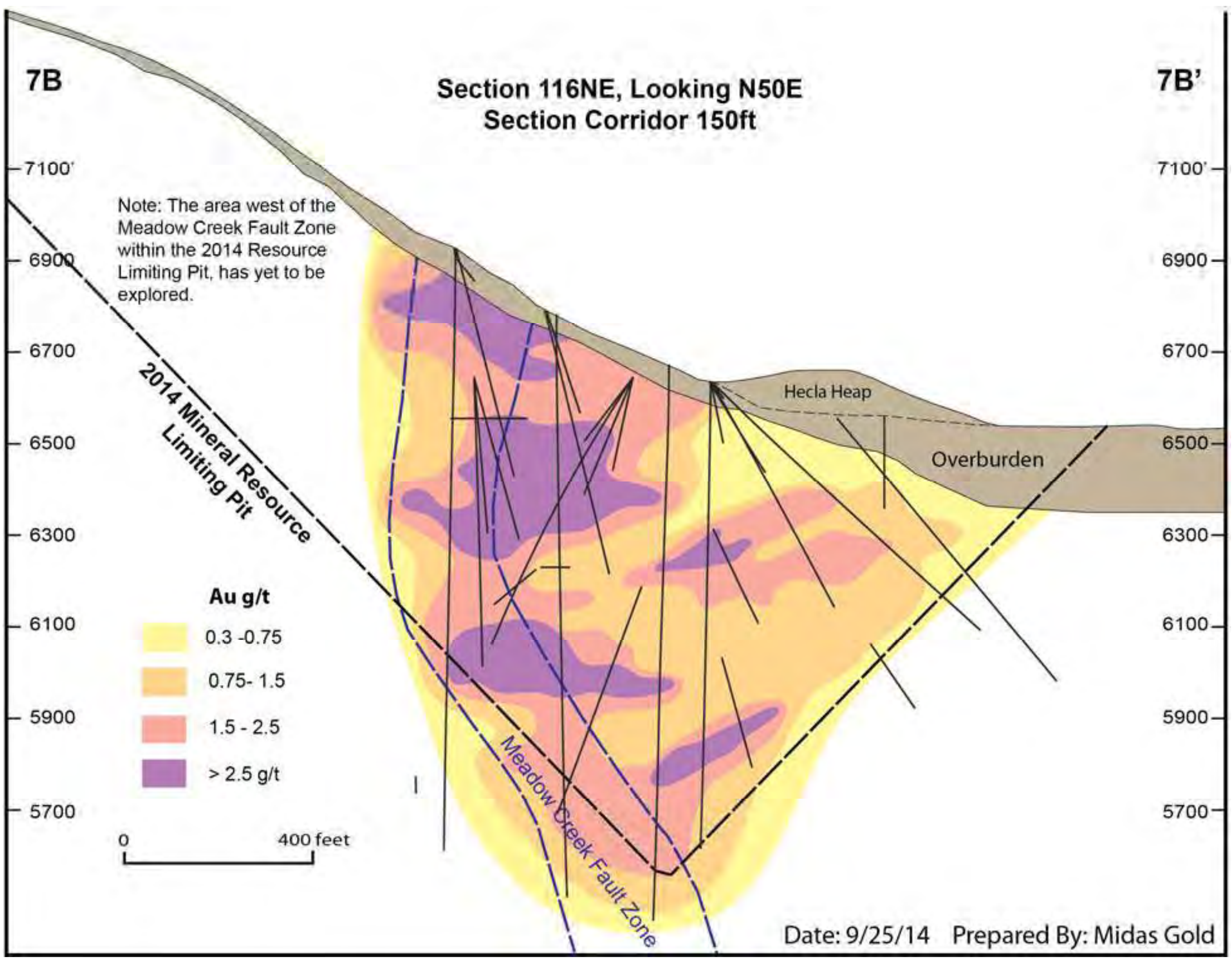
In the area of the former Homestake pit, mined by Hecla in the late 1980's and early 1990's, mineralization is more tabular and narrower than areas to the southwest (in the central domain) and is associated with multiple, north-striking, shallow west-dipping structures intersecting the main MCFZ as well as east to east-northeast striking and west to west-northwest striking, north-dipping dilatant structures.

Multiple stages of movement in the MCFZ are described in the historic literature (White, 1940; Cooper, 1951) and are evident in pit walls and in Midas Gold drill core, with the latest event marked by extensive gouge and brecciation. Various kinematic indicators suggest the latest movement involved right lateral and high angle reverse movement. The kinematics of this system have created the dilatational zones of mineralization in the Yellow Pine and Homestake pit areas. The bounding faults often contain lenses of previously mineralized material caught up in the faults during subsequent phases of deformation.

7.3.2 Hangar Flats Deposit

Mineralization in the Hangar Flats Deposit is entirely intrusive hosted, and structurally controlled and localized by the MCFZ, a generally north trending, steeply west-dipping complex fault zone with ancillary structures. The MCFZ can be traced from the main Yellow Pine Deposit south 1.85 mi through the Hangar Flats Deposit and continues for another 1.25 mi to the south, all the way to the rim of the Thunder Mountain Caldera. Past production and currently defined mineralized zones occur along variably north-plunging tabular to pipe-like bodies at the intersection of the main north-south structural feature and northeast to southwest and east to west trending steeply dipping conjugate structures and northeast trending, shallow northwest dipping (±30°) dilatant splays. The mineralized zones range in true thickness from 16 ft to over 330 ft, and can be traced several hundred feet down dip. They occur as stacked ellipsoidal lenses along the footwall to the main MCFZ which is a thick, 80-165 ft wide zone of clay gouge and heavily broken and brecciated ground. At Hangar Flats the mineralized zones become thinner, less continuous, and lower grade away from the main MCFZ (Figure 7.6).

Figure 7.6: Hangar Flats Mineralized Zone



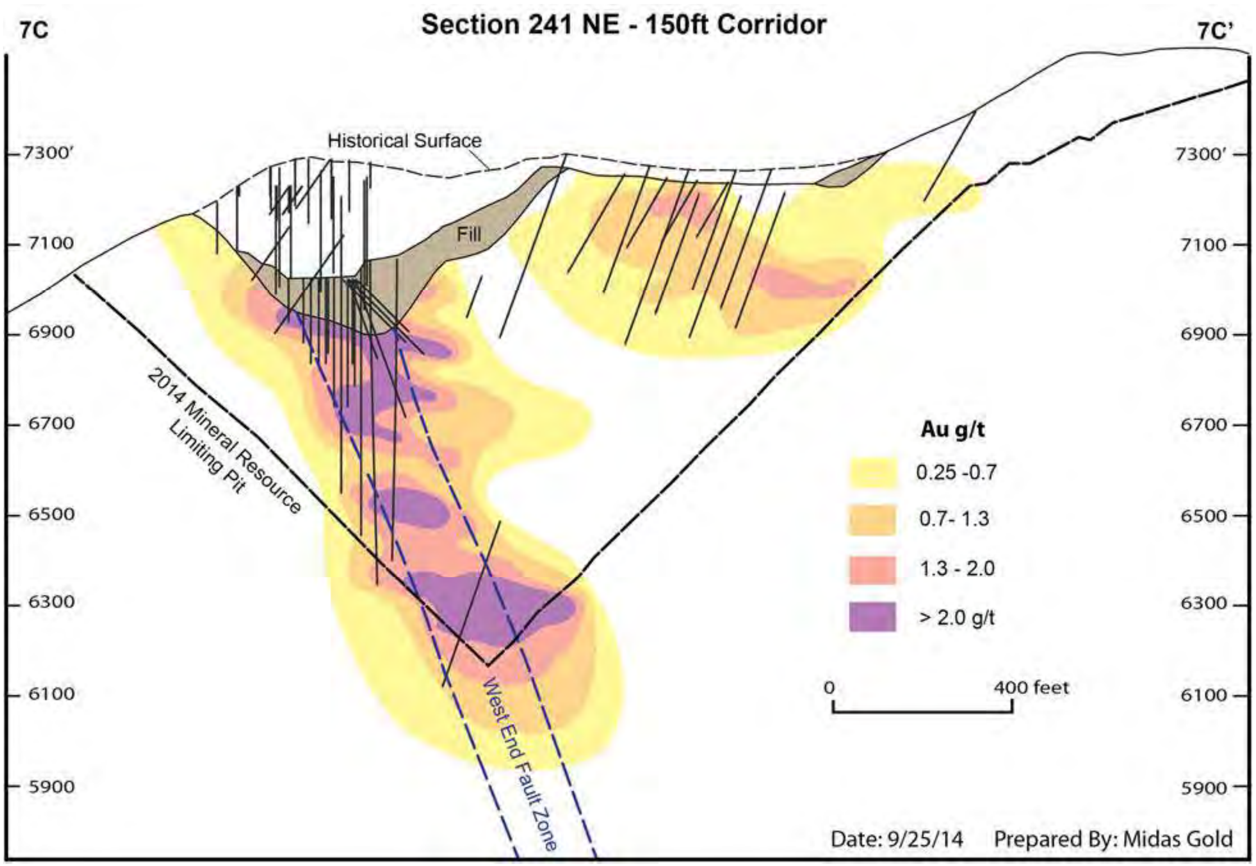
Note:
The Hangar Flats deposit was mined by underground methods between approximately 1926 and 1938. For simplicity, the underground workings are not shown on the cross section but are quantified in Section 14 of this Report.

Multiple stages of movement are described from underground mapping in the historic literature within unpublished company files and are evident in Midas Gold drill core, with the latest event marked by gouge and brecciation. Sulfide mineralized fragments have been rotated and then re-mineralized indicating several periods of movement coincident with at least some of the stages of sulfide mineralization. Various kinematic indicators suggest the latest movement along the MCFZ, at least some post-mineralization, involved right lateral and high angle reverse (i.e. west side up relative to east side) movement.

7.3.3 West End Deposit

Within the WEFZ gold mineralization occurs preferentially where the northwest-striking, northeast-dipping calc-silicate units are cut by any of the WEFZ or subsidiary faults, but all rock types host mineralization. Mineralized zones occur as stacked ellipsoidal bodies plunging along the intersection of favorable lithologic units and structural zones. True widths of these bodies range from 50 ft to over 330 ft. Drilling by Midas Gold has intersected gold mineralization associated with the WEFZ well below the historic pit bottom – as deep as 1,300 ft below the original ground surface where mineralization was exposed prior to mining. The hanging wall of the WEFZ tends to exhibit relatively more dilatant and dispersed structures relative to the footwall and, therefore, more significantly mineralized. Open space fill quartz veins are closely associated with the faults and are indicative of higher grade zones of mineralization. In addition to sulfide mineralization, open fractures along the WEFZ and subsidiary faults have allowed for oxide formation at depth from meteoric infiltration (Figure 7.7).

Figure 7.7: West End Mineralized Zone



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8 DEPOSIT TYPES

8.1 DEPOSIT MODELS

The origin of the wide variety of mineralization occurrences at Hangar Flats, West End, and Yellow Pine deposits is enigmatic and past workers have attributed the District metal endowment to deep-seated intrusives and associated deep high temperature and high pressure processes as well as shallower lower temperature, lower pressure hydrothermal processes within an epithermal environment. However, there is no single deposit model applicable to the deposits within the District that have been discovered to date. Within the Project area, the focus of past exploration and development for Au-Ag-Sb-W-Hg has been from both disseminated deposits extracted using conventional open pit methods and higher grade structurally controlled deposits extracted using various underground mining methods.

Mineralization occurs in numerous locations throughout the District in medium- to coarse-grained, felsic to intermediate intrusive host rocks and typically occurs as disseminated replacement mineralization within structurally prepared dilatant zones or adjacent to district- and regional-scale fault zones. Mineralization also occurs associated with sheeted veins, stockworks, endoskarns, and complex polymictic breccias. In the metamorphosed sedimentary rocks, mineralization occurs associated with dense fracture zones in structurally prepared sites and as stratiform manto-style replacements in reactive carbonate and calcareous siltite and schist units, as well as in cross-cutting breccia veins and dikes.

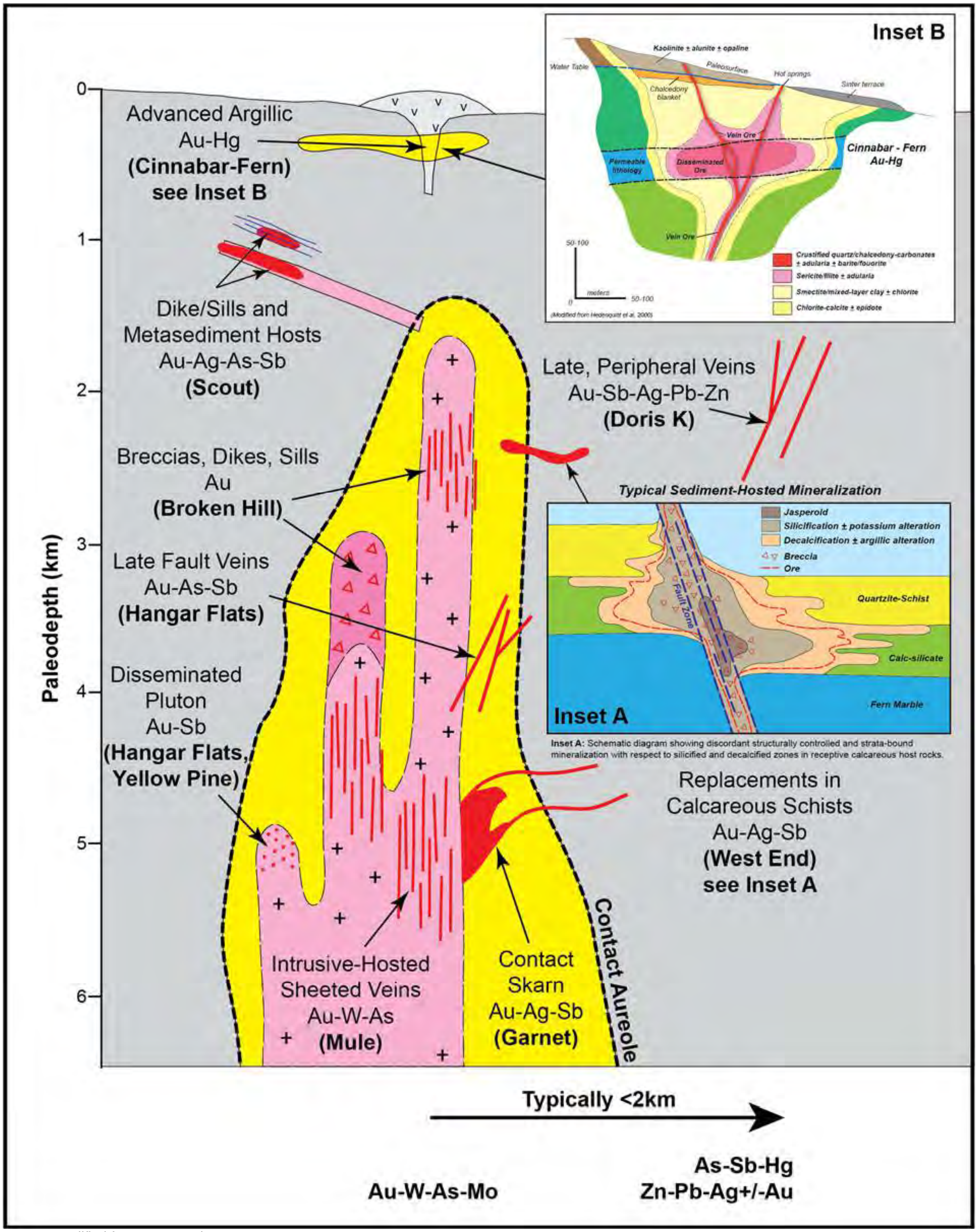
Field observations, petrographic studies, metallurgical studies, and process mineralogy studies indicate that there were likely multiple stages of mineralization, possibly separated by extended time periods. Early higher temperature, precious metal-rich mineralization with a potential magmatic fluid source was overprinted by younger, lower temperature Au and Sb-Ag mineralization; this was again overprinted by later epithermal mineralization involving meteoric water input into the hydrothermal system with a distinctly different style and geochemical signature.

The gold mineralization at the Hangar Flats and Yellow Pine deposits occurs in intrusive rocks associated with the Atlanta Lobe of the Idaho Batholith. Strong mineralization is localized along an overall north to south striking fault zone and also along northeast striking splay faults and dilatational fault jogs. Dilatant zones have generally provided conduits for movement of mineralizing hydrothermal fluids. Multiple episodes of fracturing have allowed multiple episodes of hydrothermal mineralization.

The gold mineralization at the West End Deposit occurs in metasedimentary rocks intruded by the Idaho Batholith and also within the intrusive rocks. The metasediments occur as pendants and xenoliths within the intrusive rocks. Strong mineralization is localized along a northeast striking fault zone and splay faults that strike northeast and east. Pull-apart fracturing along dilatant northeast fault jogs and splays provided conduits for movement of mineralizing hydrothermal fluids. Multiple episodes of fracturing allowed multiple episodes of hydrothermal mineralization.

A schematic of the geologic setting for the various deposits and exploration prospects is shown on Figure 8.1. Based on the nature and scale of the hydrothermal alteration systems present, the deposits are interpreted to be related to intrusive activity. This figure (modified from Lang et al., 2000) illustrates the spatial relationships of each major deposit type, the intrusion(s), and the associated hydrothermal systems.

Figure 8.1 Mineralization Model for Intrusive-Related Gold Systems in the Stibnite Mining District



Source: Modified from Lang et al., 2000

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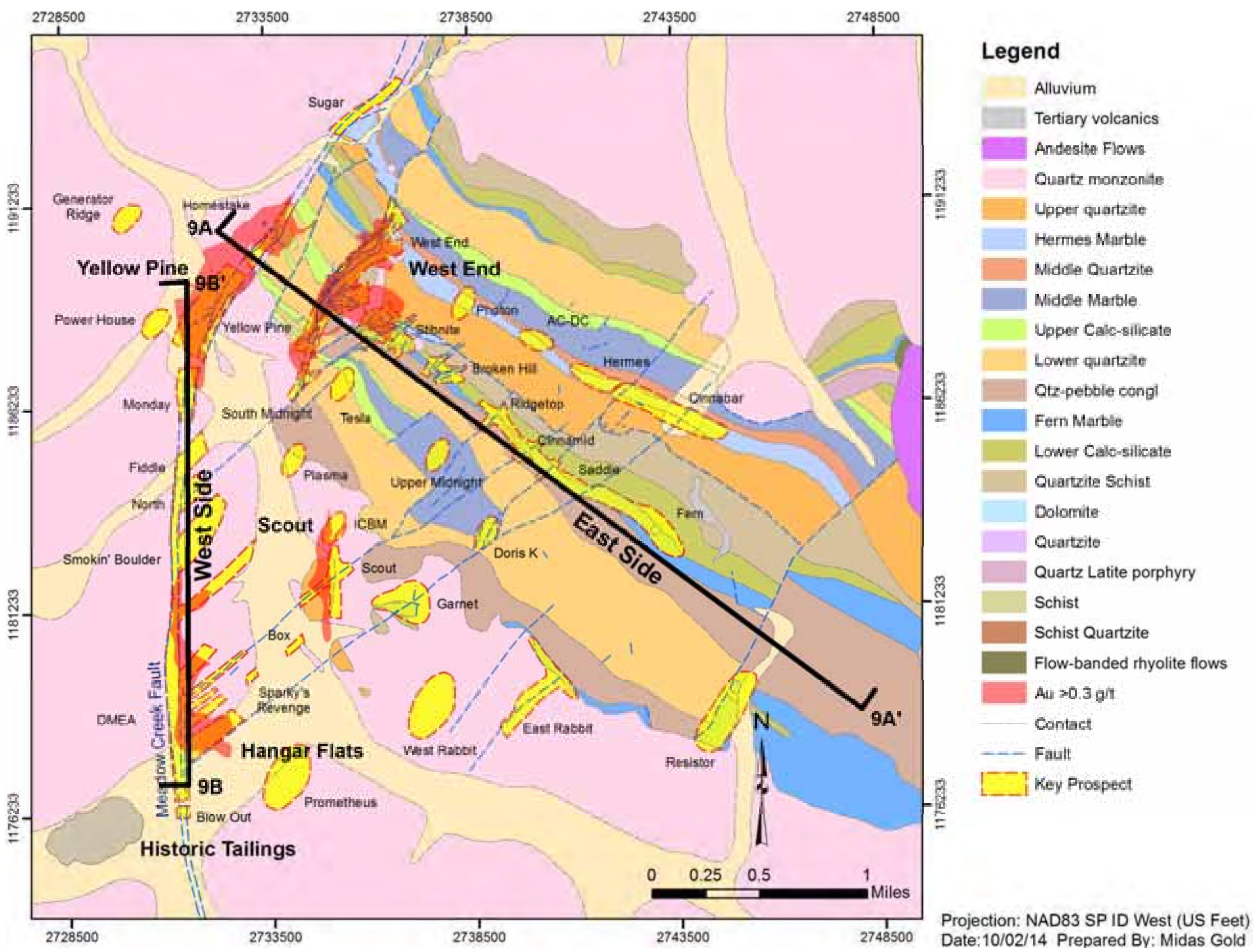
9 EXPLORATION

9.1 EXPLORATION POTENTIAL

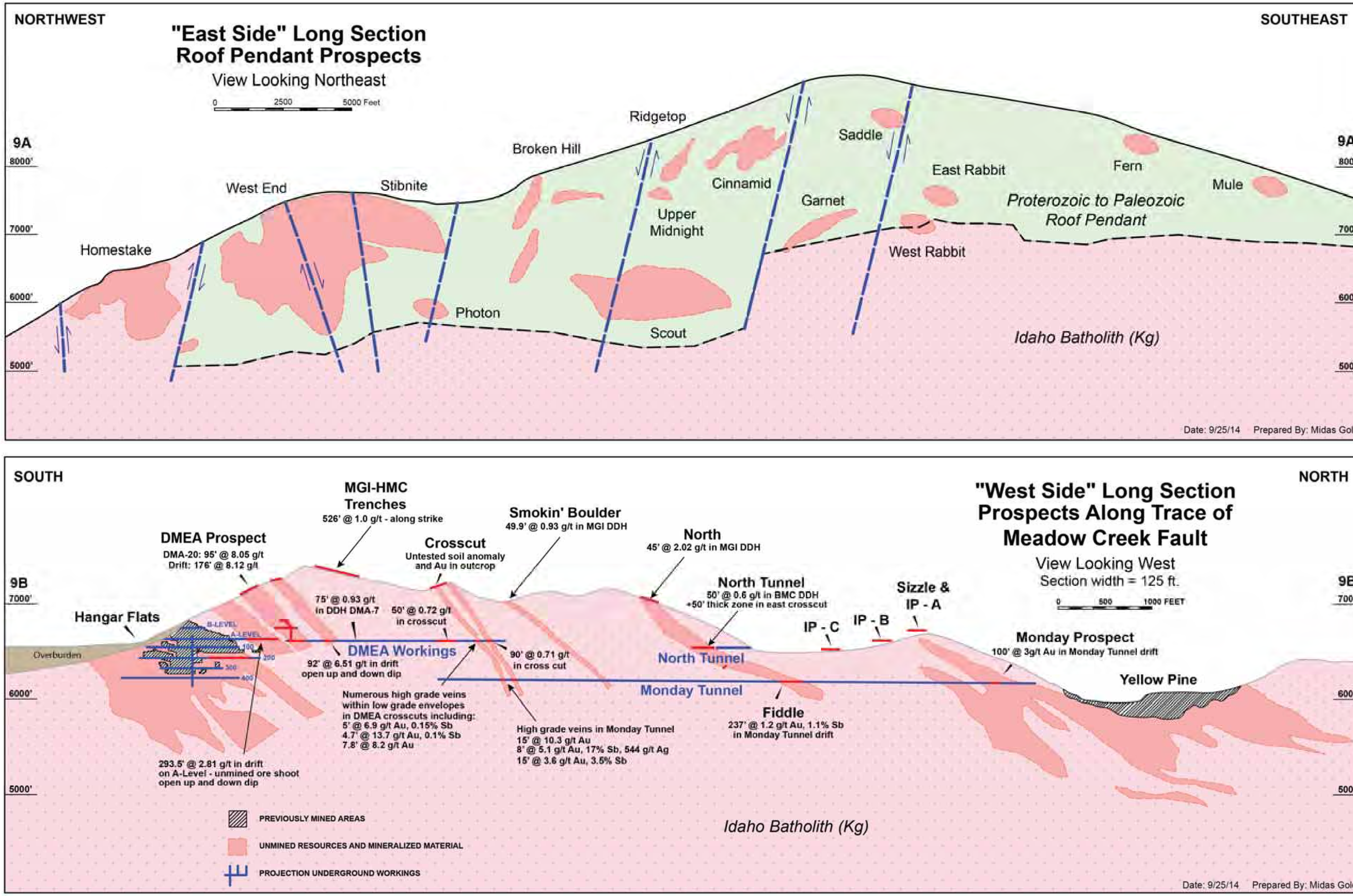
Numerous prospects have been discovered during exploration and development activities in the Districts over the past nearly 100 years using a variety of methods; some of these prospects were developed into mines while others remain undeveloped. Besides pit expansion possibilities around the main deposits, other exploration targets may one day warrant consideration for development if they can be proven viable after additional exploration, environmental, socio-economic, metallurgical, engineering, and other appropriate studies. Midas Gold has developed an extensive pipeline of exploration targets, which are summarized below and shown on Figure 9.1. The long sections associated with the District geology map are shown in Figure 9.2.

Figure 9.1: District Geology Map Showing Regional Prospects



The exploration targets discussed herein include more advanced prospects that have had past production and/or adequate drilling to infer good potential for high grade mineralization that might be exploited via underground mining methods (such as Scout and Garnet). In addition less advanced, but still promising underground prospects (Upper Midnight and Doris K) that have received less drilling, still have strong indications of potential high-grade mineralization. There are also more advanced prospects that have had enough drilling to infer good potential for disseminated mineralization that might be exploited via bulk tonnage, open pit mining methods. These prospects are located along the Broken Hill-Saddle trend, where past exploration focused on the search for leachable oxide ores.

Figure 9.2: East Side and West Side Long Sections through the District



Other areas with potential for new discoveries lie between the known deposits, such as the string of prospects that lie between the Yellow Pine and Hangar Flats deposits namely, Monday Tunnel, Sizzle, Fiddle, North, Smokin' Boulder and North DMEA. Midas Gold has also delineated several new prospects that have had little systematic work or were only recently discovered namely, Mule, Volt, East and West Rabbit and others.

Exploration data for the target areas discussed above and below include geophysical data, geochemistry from soil, rock and trench samples, and results from widely spaced drill holes; as a result the potential size and tenor of the targets are conceptual in nature. There has been insufficient exploration to define mineral resources on these prospects and this data may not be indicative of the occurrence of a mineral deposit. Such results do not provide assurance that further work will establish sufficient grade, continuity, metallurgical characteristics and economic potential to be classed as a category of mineral resource.

9.2 GRIDS AND SURVEYS

Numerous local grids have been used on site since the 1920s and control points have been re-established where possible and practical. Original errors in earlier surveys are known to exist, but most data have been found to be reliable to within approximately 6-10 ft. Some datasets are less reliable, particularly historic soil and ground geophysical surveys. Midas Gold completed topographic and aerial photographic surveys in 2009 and 2012 for geodetic control. All Midas Gold drill locations are surveyed with survey grade instruments and typically have a level of precision of $\pm$ 10 inches. Rock and soil samples are usually surveyed with hand held GPS instruments and have been determined to usually be reliable to within ±10-20 ft.

9.3 GEOLOGIC MAPPING

The Project area has been mapped by numerous past workers and by Midas Gold staff. Mapping was typically completed by past operators along many roads, previously operating open pits and underground workings that are now reclaimed, covered and/or inaccessible, as well as in cross country traverses. Where possible and practical, map data have been field checked by Midas Gold geologists and found to be reliable for the needs of the Project. Midas Gold staff have remapped areas, where needed, to obtain additional information.

9.4 GEOCHEMICAL SAMPLING

Past operators collected and analyzed thousands of soils, rock chips, underground channel, surface chip, trench and drill hole samples utilizing a variety of laboratories and methods. Not all sample information is fully documented with chain of custody, lab methods and/or QA/QC. However, the geochemical data are considered reliable enough to utilize for basic exploration purposes. In areas of exploration interest, Midas Gold staff have collected samples using current industry standard protocols and certified analytical laboratories to verify past work and/or expand upon it.

9.5 GEOPHYSICS

Midas Gold contractors completed a helicopter-supported 222 line-mi aeromagnetics survey in 2009, covering 33 mi^2, followed by a more detailed 595 line-mi airborne electromagnetic (EM) and magnetics survey covering a larger area in 2011. The data was filtered, gridded, post-processed and integrated with geologic and geochemical data to generate and evaluate target areas. Contractors also completed induced polarization – resistivity surveys (IP) along 13 lines, totaling 13 line-mi, and Controlled Source Audio Magnetotellurics surveys (CSAMT) along 13 lines totaling 31 line-mi over the central part of the District. Numerous, high quality anomalies were identified and indicate a large area of anomalous IP and CSAMT responses between the Yellow Pine and Hangar Flats deposits, as well as in other areas.

9.6 PETROLOGY, MINERALOGY AND RESEARCH

Extensive characterization of mineralogy has been completed as part of Midas Gold's metallurgical and mineralization characterization testing program using conventional petrographic and near infrared spectrometry methods, as well as Quantitative Evaluation of Minerals by Scanning (QEMSCAN) electron microscopy. The Idaho Geologic Survey has been conducting radiometric dating studies of intrusive- and metasedimentary-hosted mineralization to provide information on the approximate ages of mineralization and detrital zircon studies to evaluate age and provenance of metasedimentary rocks. This research is ongoing and has not yet been published, but preliminary data have been made available to Midas Gold.

9.7 POTENTIAL FOR EXPANSION OF THE YELLOW PINE, HANGER FLATS AND WEST END DEPOSITS

All three major deposits with Mineral Resources reported herein remain open to expansion and potential is described in the following sections.

9.7.1 Yellow Pine

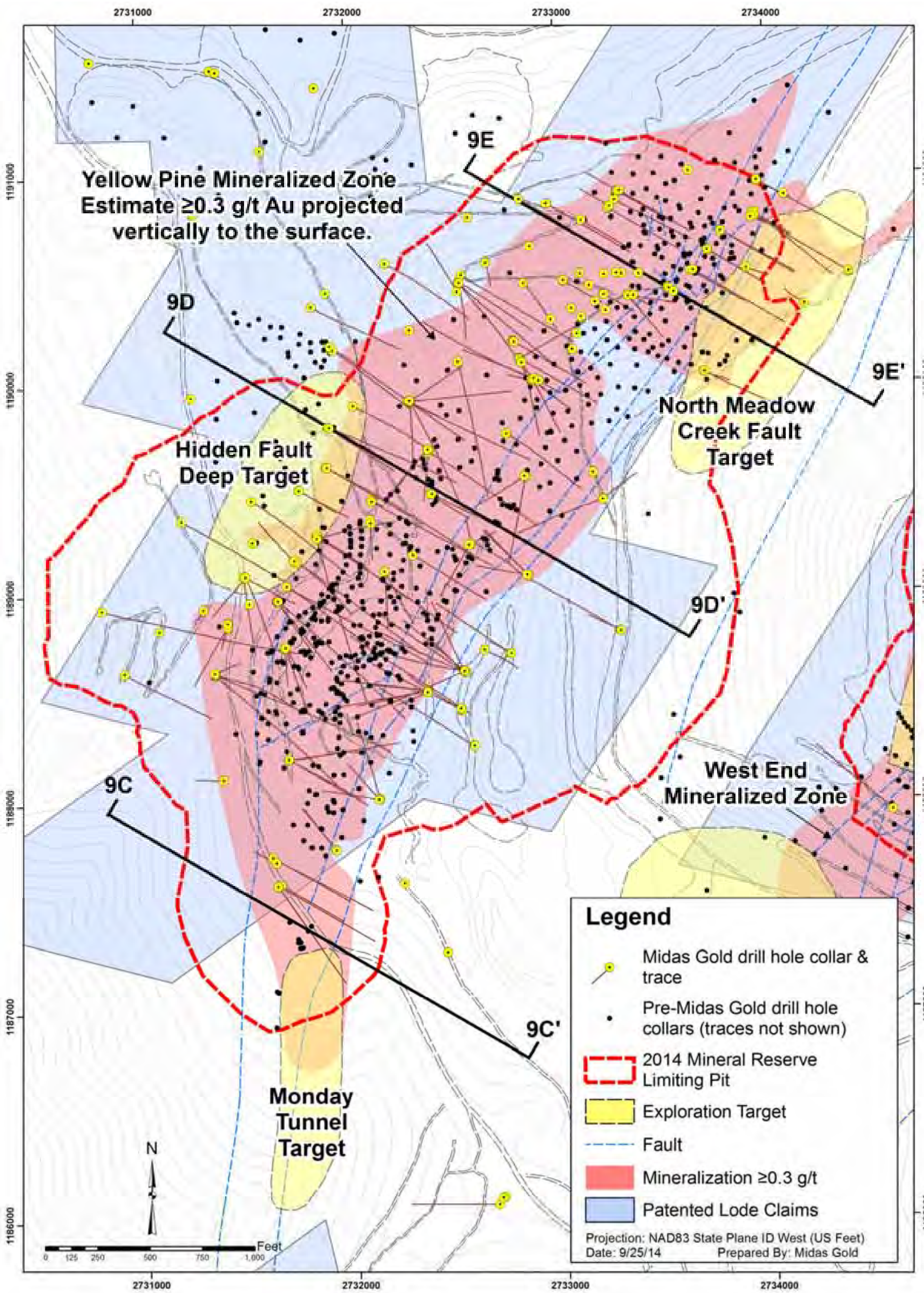
The Yellow Pine Deposit is open at depth and along strike in the north, northeast and southwest directions (Figure 9.3). Targets are defined by mineralized holes drilled by both Midas Gold and pre-Midas Gold operators. Highlights of some of the holes defining these targets are tabulated in Table 9.1 and the areas shown on Figure 9.3.

Table 9.1: Significant Drill Intercepts within the Yellow Pine Expansion Targets

Target	Operator	Drill Hole ID	Collar Dip (˚)	Collar Azimuth (˚)	From (ft)	To (ft)	Interval (ft)	Gold(1,2) (g/t)
Hidden Fault Deep Zone	Midas Gold	MGI-11-187	-72.5	120	633	964	331	1.24
Hidden Fault Deep Zone	Midas Gold	MGI-12-224	-79	120	560	766	216	1.21
North Meadow Creek Fault	Bradley Mining Co.	B-043	-45	105	140	185	45	5.86
North Meadow Creek Fault	Hecla Mining Co.	89-02GT	-45	122	200	320	120	1.57
North Meadow Creek Fault	Midas Gold	MGI-11-082	-50	117	498	632	134	1.89
North Meadow Creek Fault	Midas Gold	MGI-13-307	-60	130	660	822	162	5.42
Monday Tunnel	Midas Gold	MGI-11-140	-50	145	300	395	95	2.80

Note:
(1) *Selected intercepts composited with length weighted averages with cut-off grade of 0.75 g/t Au and/or 0.3% Sb reported, >10 ft composite length and <10 ft of internal waste below 0.5 g/t Au and/or 0.1% Sb.*
(2) *All gold and silver grades denoted g/t herein and in subsequent sections of this Report are reported in units of grams per metric tonne. Grades denoted oz/st are reported in units of troy ounces per short ton.*

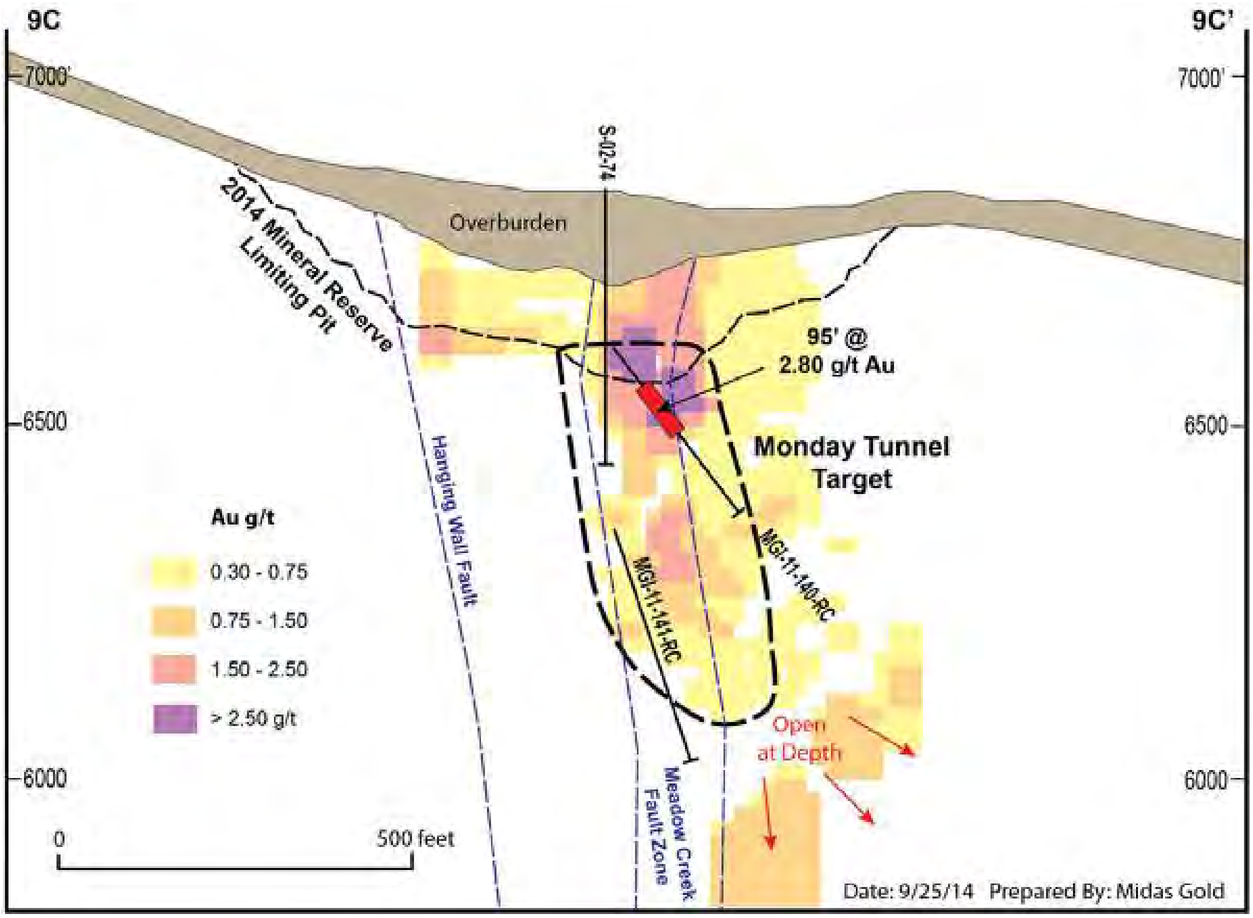
Figure 9.3: Plan Map of Yellow Pine Showing Potential Expansion Targets



Monday Tunnel Target

The continuation of the Meadow Creek Fault, south of the main Yellow Pine Deposit, has been the subject of limited drilling by Bradley Mining Company and Midas Gold. The most significant Midas Gold intercept in this area returned 95 ft averaging 2.80 g/t gold below the 2014 Mineral Reserve Limiting Pit (Figure 9.4) and the zone remains open along strike and down dip.

Figure 9.4: Cross Section of the Monday Tunnel Target with an 80 ft Corridor

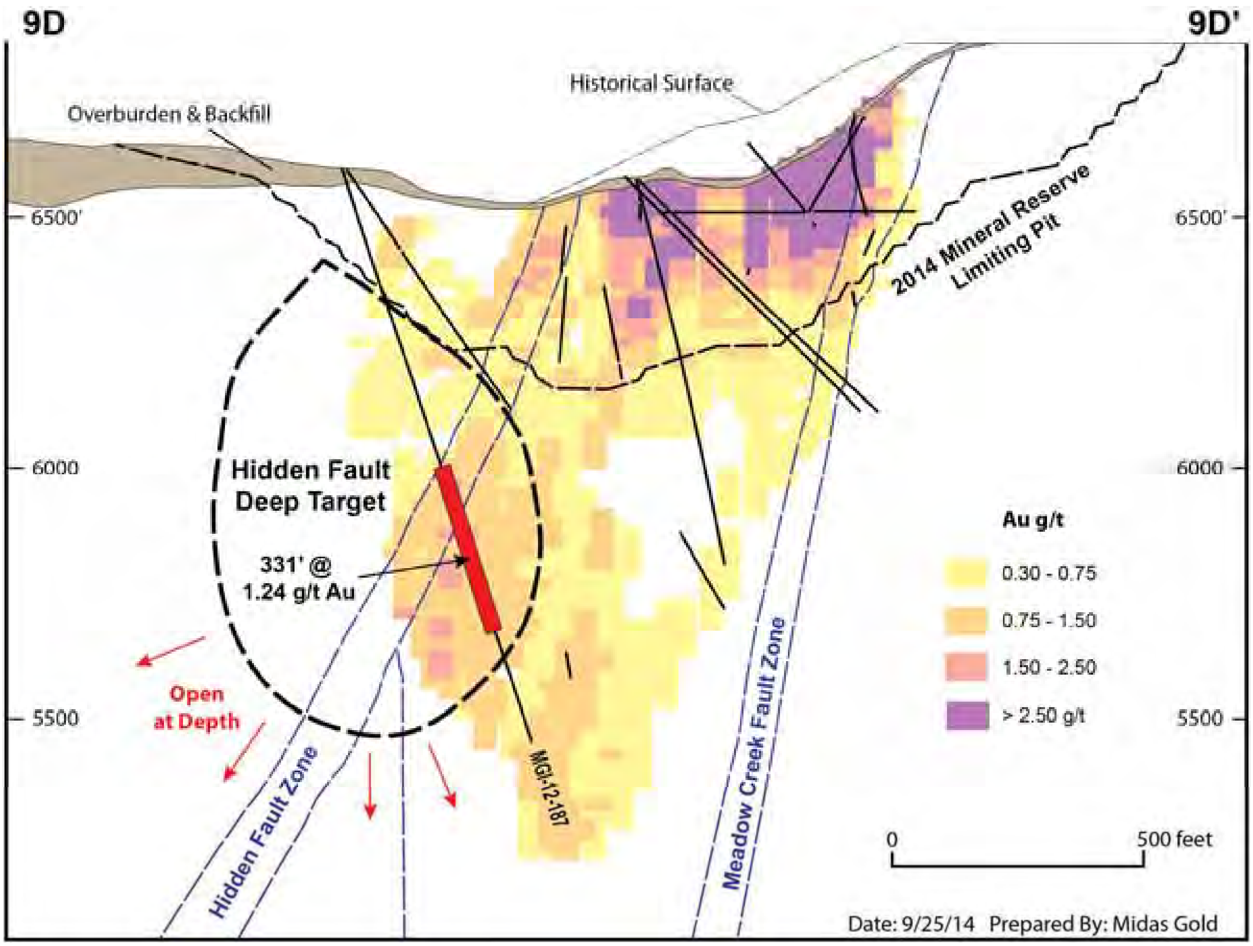


Note:
Potential mineralization reported here as a prospect may be partially included within the mineral resources discussed in Section 14 of this Report.

Hidden Fault Deep Target

This area is located at the northwest edge of the Yellow Pine Deposit (Figure 9.5), along the trace of the Hidden Fault. The most significant Midas Gold intercept returned 331 ft averaging 1.24 g/t Au (Figure 9.5) and the zone remains open.

Figure 9.5: Cross Section of the Hidden Deep Fault Target with an 80 ft Corridor

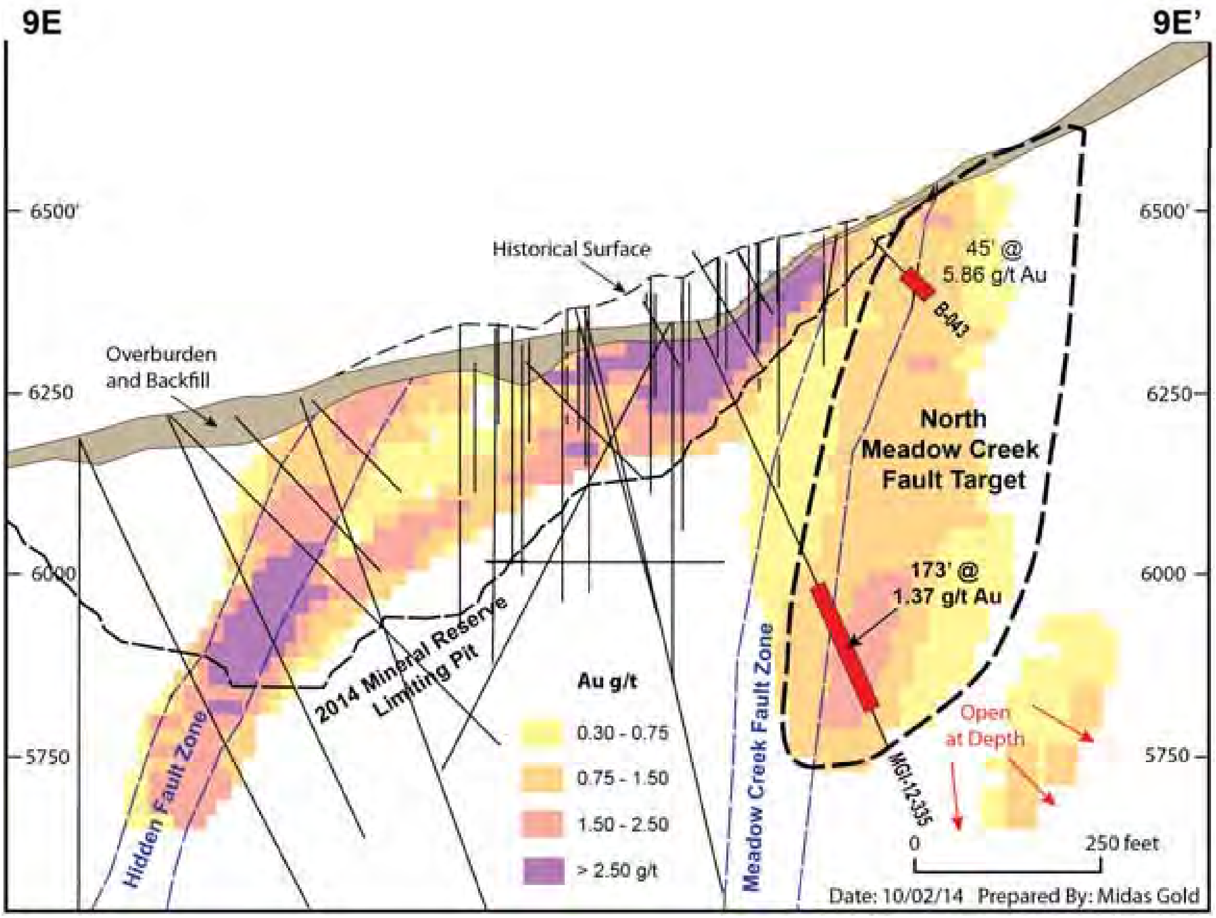


Note:
Potential mineralization reported here as a prospect may be partially included within the mineral resources discussed in Section 14 of this Report.

North Meadow Creek Fault Target

This target lies on the northeast side of the Yellow Pine Deposit and is defined by several holes drilled by pre-Midas Gold operators and Midas Gold (Figure 9.6). Significant drill intercepts include 45 ft averaging 5.86 g/t gold and 173 ft averaging 1.37 g/t gold.

Figure 9.6: Cross Section of the North Meadow Creek Fault Target with a 250 ft Corridor



Note:
Potential mineralization reported here as a prospect may be partially included within our resources discussed in Section 14 of this Report.

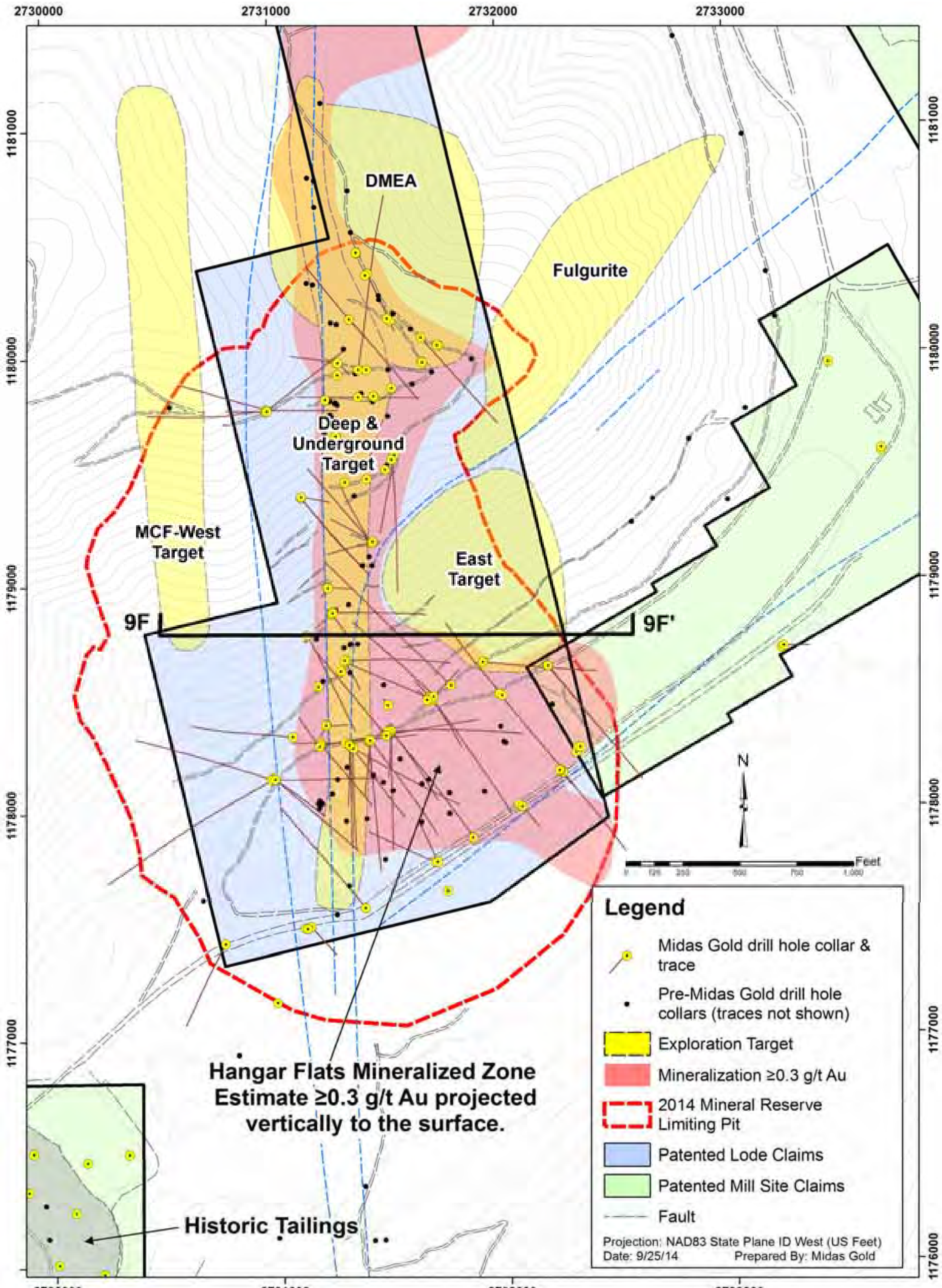
9.7.2 Hangar Flats

The Hangar Flats Deposit formed along the Meadow Creek Fault zone (MCFZ) and the 3,000(+) ft long corridor north, east and west of the main deposit is inadequately drill-tested outside of the known deposit (Figure 9.7).

Hangar Flats Deep Target

Historic sampling and production records from the former Meadow Creek Mine define a zone of high grade gold-antimony mineralization in a 30-330 ft wide corridor along the eastern boundary of the MCFZ that remains open along strike and down dip. Figure 9.8 shows drill hole MGI-12-203, which intersected multiple high grade intercepts, the most significant included 121 ft grading 2.96 g/t Au.

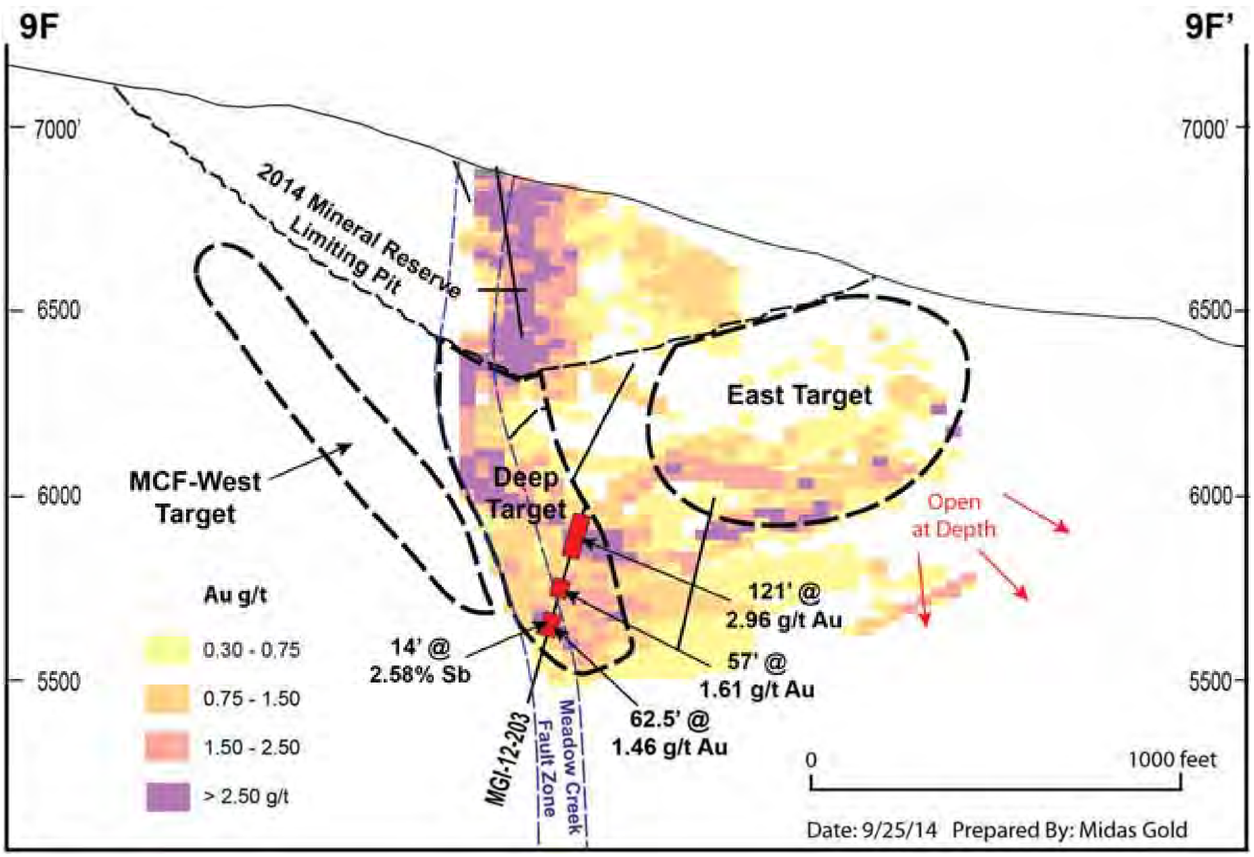
Figure 9.7: Plan map Showing the Hangar Flats Expansion Targets



DMEA Workings Target

This target lies beneath the northern part of the Hangar Flats PFS Mineral Reserve Limiting Pit and represents the northern extension of the "Hangar Flats Deep Target". The MCFZ is poorly tested over a distance of at least 1,000 ft at DMEA, which has been explored by a network of tunnels driven along the MCFZ. The underground workings were extensively mapped and sampled in the 1950s, which indicated the presence of north-east trending high grade vein systems. Pre-Midas Gold underground channel samples, from crosscuts, reportedly intersected a large zone of mineralization with a length-weighted average grade of 6.5 g/t Au over 92 ft and 1.56 g/t Au over 300 ft collected perpendicular to the MCFZ, while underground drill holes intersected significant high grade intercepts (Table 9.2).

Figure 9.8: Cross Section of Hangar Flats with a 160 ft Corridor



Note:
Potential mineralization reported here as prospects may be partially included within our resources discussed in Section 14 of this Report.

Table 9.2: Significant Drill Intercepts within the Hangar Flats Expansion Targets

Target	Operator	Drill Hole ID	Collar Dip (˚)	Collar Azimuth (˚)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Antimony (%)
Hangar Flats Deep	Midas Gold	MGI-11-058	-90	-	588	713	125	3.13	1.45(1)
Hangar Flats Deep	Midas Gold	MGI-12-193	-88	324	921	1170	249	1.55	2.54
Hangar Flats Deep	Midas Gold	MGI-12-165	-79	320	825	870	45	1.40	1.20(2)
					915	960	45	1.29	1.12(3)
Hangar Flats Deep	Midas Gold	MGI-11-103	-70	140	605	644	39	4.84	-
					695	847	135	3.51	-

Target	Operator	Drill Hole ID	Collar Dip (˚)	Collar Azimuth (˚)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Antimony (%)
Hangar Flats Deep	Midas Gold	MGI-12-203	-65	320	696	817	121	2.96	
					912.5	970	57.5	1.61	0.26(4)
					1012	1074.5	62.5	1.46	2.58(5)
DMEA Workings	Midas Gold	MGI-12-331	-89	291	334	418	84	3.65	-
					441	480	39	1.36	0.10
					551	584	33	1.05	-
					657	696	39	2.71	-
DMEA Workings	Midas Gold	MGI-09-07	-70	90	679	836	157	5.09	0.30
					853	926	73	4.89	-
DMEA Workings (UG)(6)	Bradley Mining	DMA-20	-44	320	10	135	125	6.62	0.51(7)
DMEA Workings	Midas Gold	MGI-11-080	-90	-	619	654	35	2.61	-
DMEA Workings	Midas Gold	MGI-12-181	-90	-	520	560	40	2.66	-
DMEA Workings	Midas Gold	MGI-12-197	-90	-	838	889	51	2.59	-
					940	971	31	1.58	-
Underground Drill Target	Midas Gold	MGI-12-192	-83	280	344	434	90	2.88	-
					543	612	69	1.03	0.09
					1006	1300	294	1.57	2.76
MCFZ West	Midas Gold	MGI-11-099	-75	310	1507	1659	152	1.34	0.83(8)

Notes:
(1) Sb over 83.5 foot interval.
(2) Sb over 25 foot interval.
(3) Sb over 40 foot interval.
(4) Sb over 17.5 foot interval.
(5) Sb over 14 foot interval.
(6) UG = Underground drill hole.
(7) Sb over 95 foot interval.
(8) Sb over 115 foot interval.

Underground Drill Target

High grade gold-antimony mineralization has been intersected over a 2,000 ft strike length and over a 1,000 ft vertical extent and remains open to expansion at depth. One of the more significant intercepts in this target area, cut in drill hole MGI-12-192, included 294 ft grading 1.57 g/t Au and 2.76% Sb illustrating this potential (Table 9.2).

MCFZ West Target

A geotechnical hole (MGI-11-099), drilled across the western limits of the conceptual PEA pit, intercepted significant mineralization (Table 9.2) and, based on geophysical surveys and oriented core data, mineralization is interpreted to be open and possibly extends along strike and up and down dip (Figure 9.8).

HF East Target

A large area east of the Hangar Flats Deposit has not been drill tested and is open to expansion. The area is underlain by mineralized low angle faults exposed east of the deposit which are known to control mineralization in the main portion of the Hangar Flats Deposit (Figure 9.8).

9.7.3 West End

There is potential to expand the West End Deposit at depth and along strike to the northeast and southwest. Highlights of significant drill intercepts from these areas are listed in Table 9.3 and the areas shown on Figure 9.9.

Figure 9.9: Plan Map of West End Showing Potential Expansion Zones

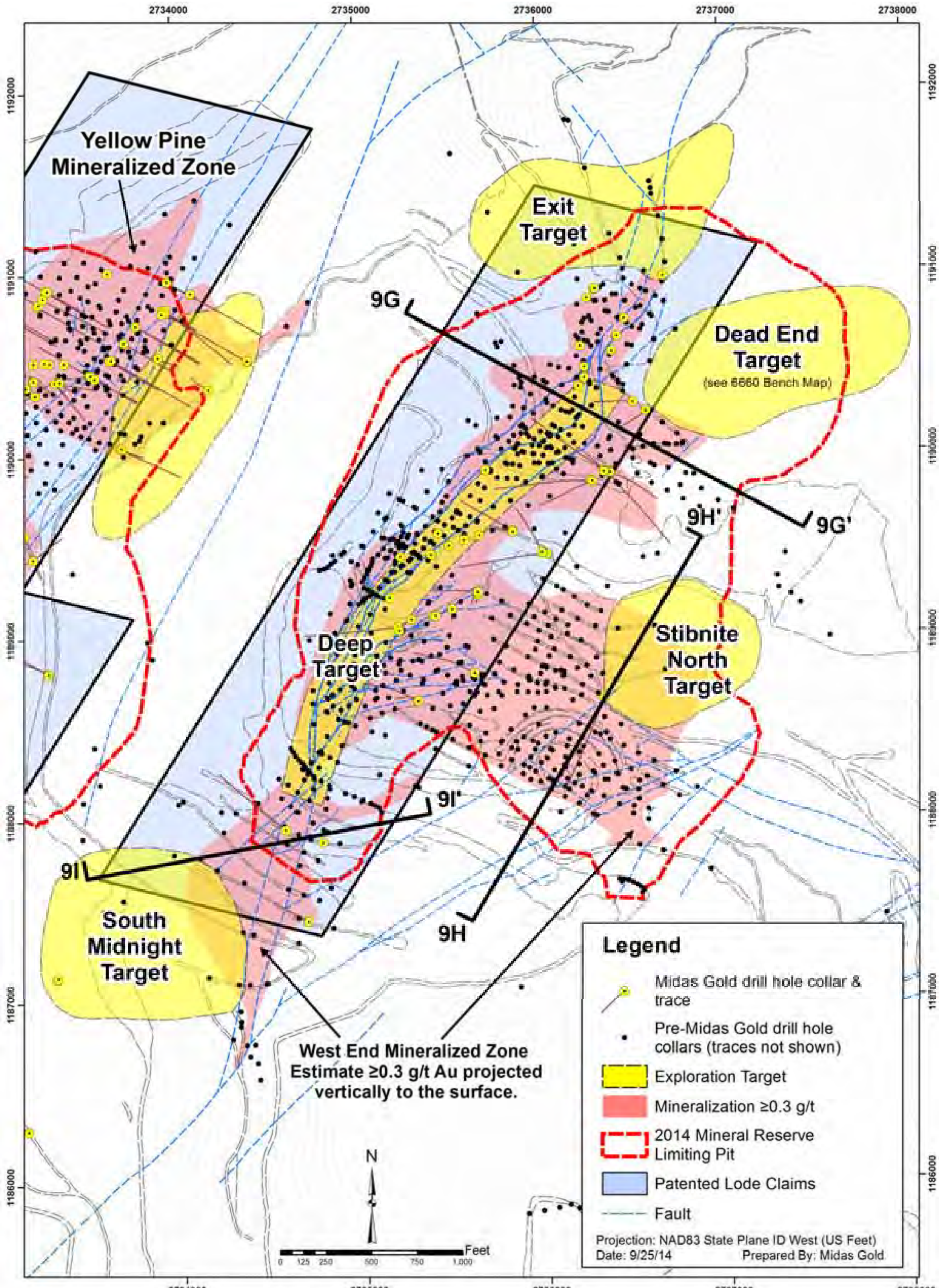


Table 9.3: Significant Drill Intercepts within the West End Expansion Targets

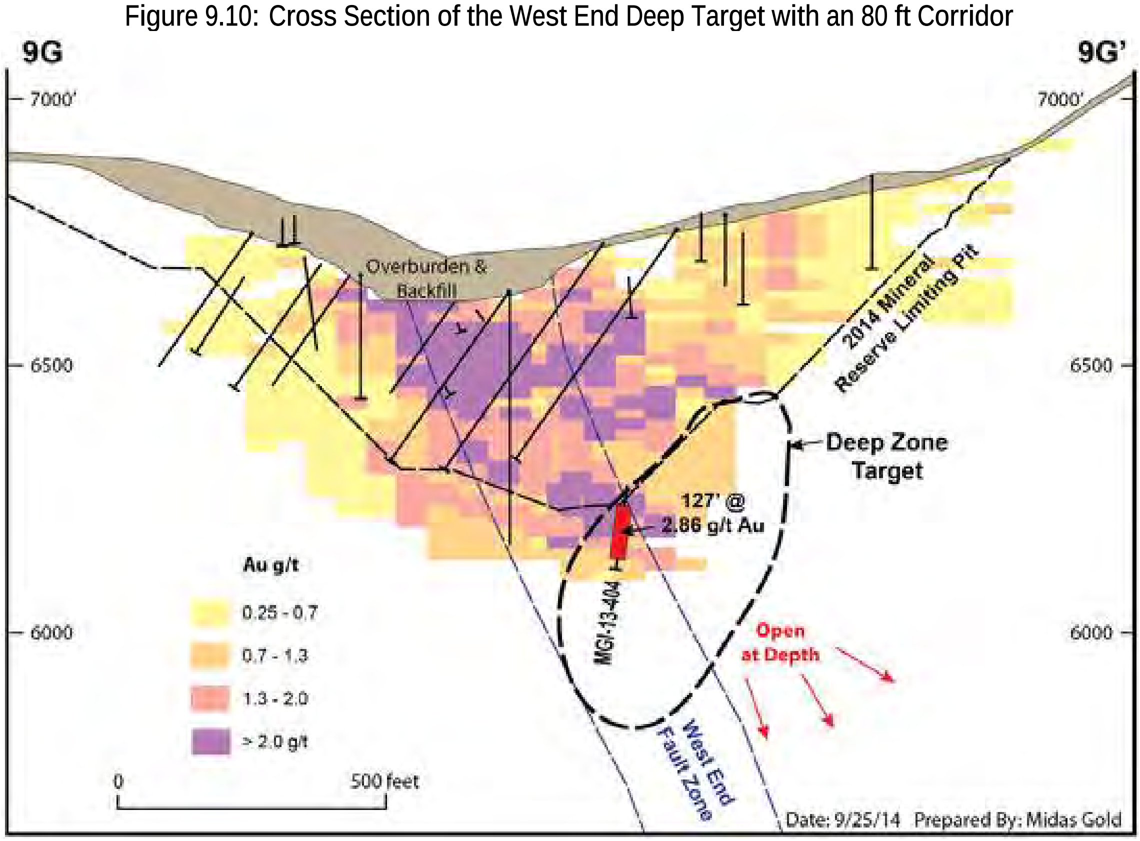
Target	Operator	Drill Hole ID	Collar Dip (˚)	Collar Azimuth (˚)	From (ft)	To (ft)	Interval (ft)	Gold(1) (g/t)
West End Deep	Midas Gold	MGI-13-404	-75	230	521	648	127	2.86
West End Deep	Midas Gold	MGI-11-139	-66	260	835	1065	230	2.27
West End Deep	Midas Gold	MGI-12-290	-71	298	762	906	144	1.07
Dead End	Pioneer	W-110	-55	80	125	180	55	1.04
Dead End	Superior	WER83-23	-90	NA	110	340	230	1.10
Stibnite North	Pioneer	89-78	-70	300	250	405	155	3.46
Stibnite North	Midas Gold	MGI-10-37	-45	202.5	418	615	197	1.79
Stibnite North	Pioneer	89-57	-70	300	415	510	95	3.16
Stibnite North	Pioneer	89-75	-70	300	460	630	170	1.4

Note:
(1) Selected intercepts composited with length weighted averages with cut-off grade of 0.75 g/t Au and/or 0.3% Sb reported, >10 ft composite length and <10 ft of internal waste below 0.5 g/t Au and/or 0.1% Sb.

West End Deep Target

This target consists of a poorly explored area 330 ft wide and extending approximately 2,100 ft along strike beneath the Mineral Reserve Limiting Pit (Figure 9.10). The area is defined by three Midas Gold drill holes, with the most significant intercepts being 2.25 g/t Au over 230 ft and 2.86 g/t Au over 127 ft in MGI-11-139 and MGI-13-404, respectively.

Figure 9.10: Cross Section of the West End Deep Target with an 80 ft Corridor

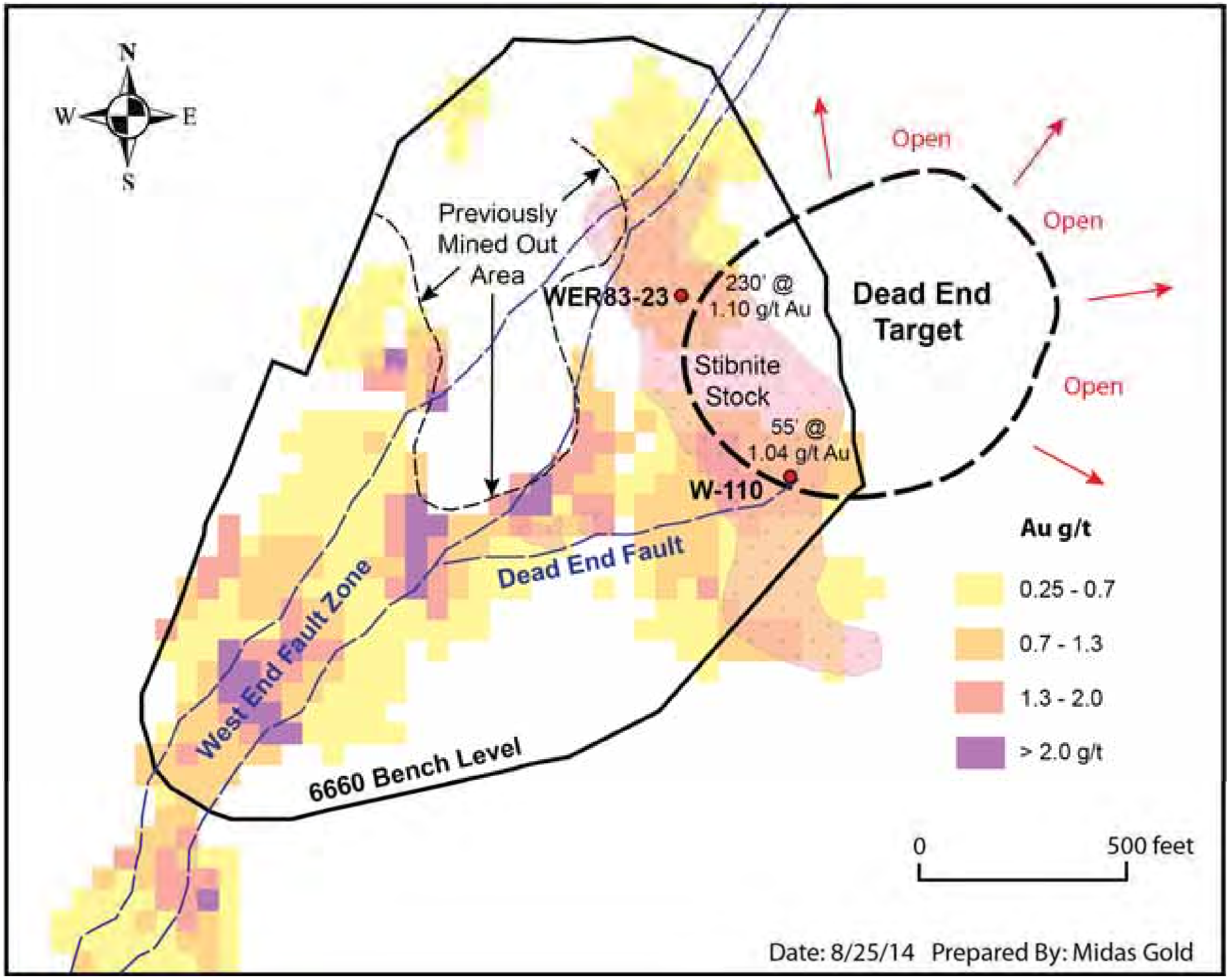


Note:
Potential mineralization reported here as a prospect may be partially included within the mineral resources discussed in Section 14 of this Report.

Dead End Fault Target

This target is approximately 260 ft wide and extends over a strike length of 1,000 ft. The target is defined by holes drilled by pre-Midas Gold operators, with the most significant intercept being 230 ft with an average grade of 1.1 g/t Au along the east-northeast striking Dead End Fault that originates at a bend in the West End Fault system (Figure 9.11).

Figure 9.11: Level Section of the Dead End Fault Target with a 30 ft Thickness

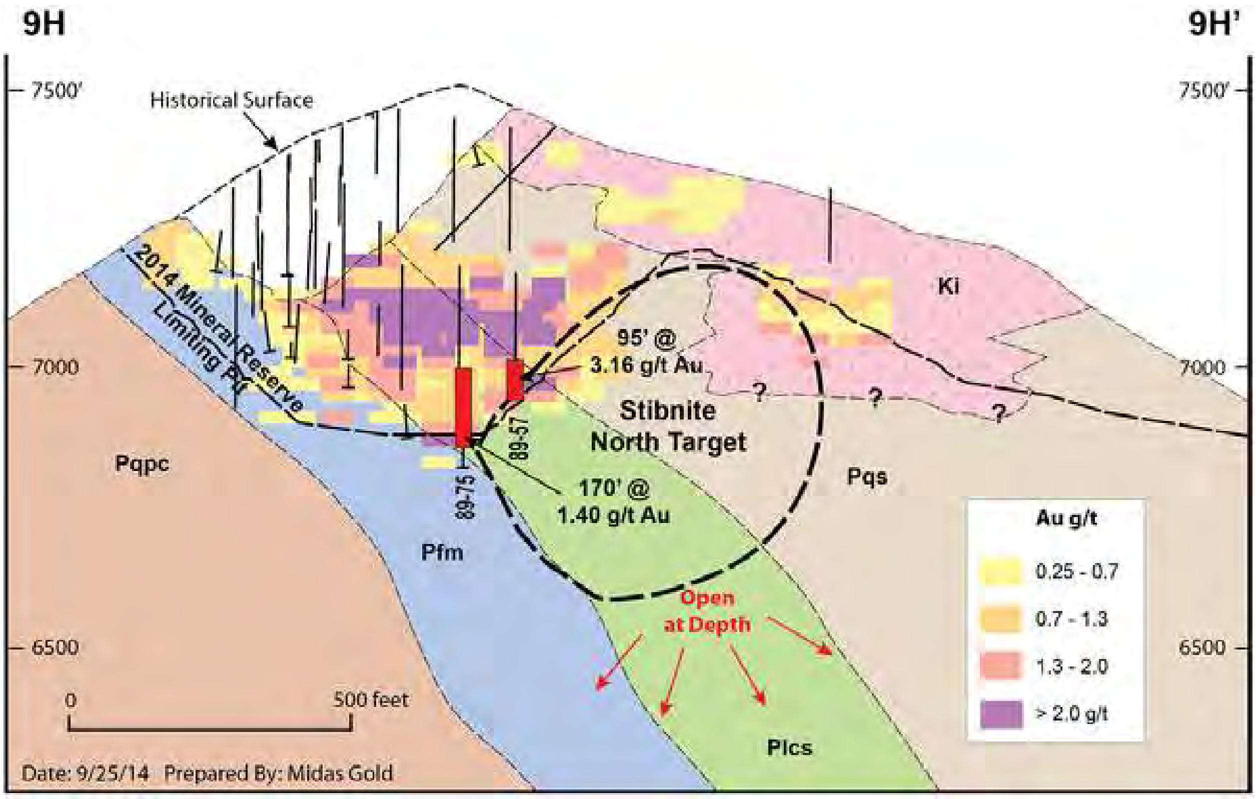


Note:
Potential mineralization reported here as a prospect may be partially included within the mineral resources discussed in Section 14 of this Report.

Stibnite North Target

The Stibnite North target is defined by Midas Gold and Pioneer Metals drill holes, with the most significant intercept being 155 ft with an average grade of 3.46 g/t Au in hole 89-78 (Table 9.3 and Figure 9.12). Mineralization may continue down dip and along strike within favorable faults and lithologies extending past the Mineral Reserve Limiting Pit.

Figure 9.12: Cross Section of the Stibnite North Target with an 80 ft Corridor

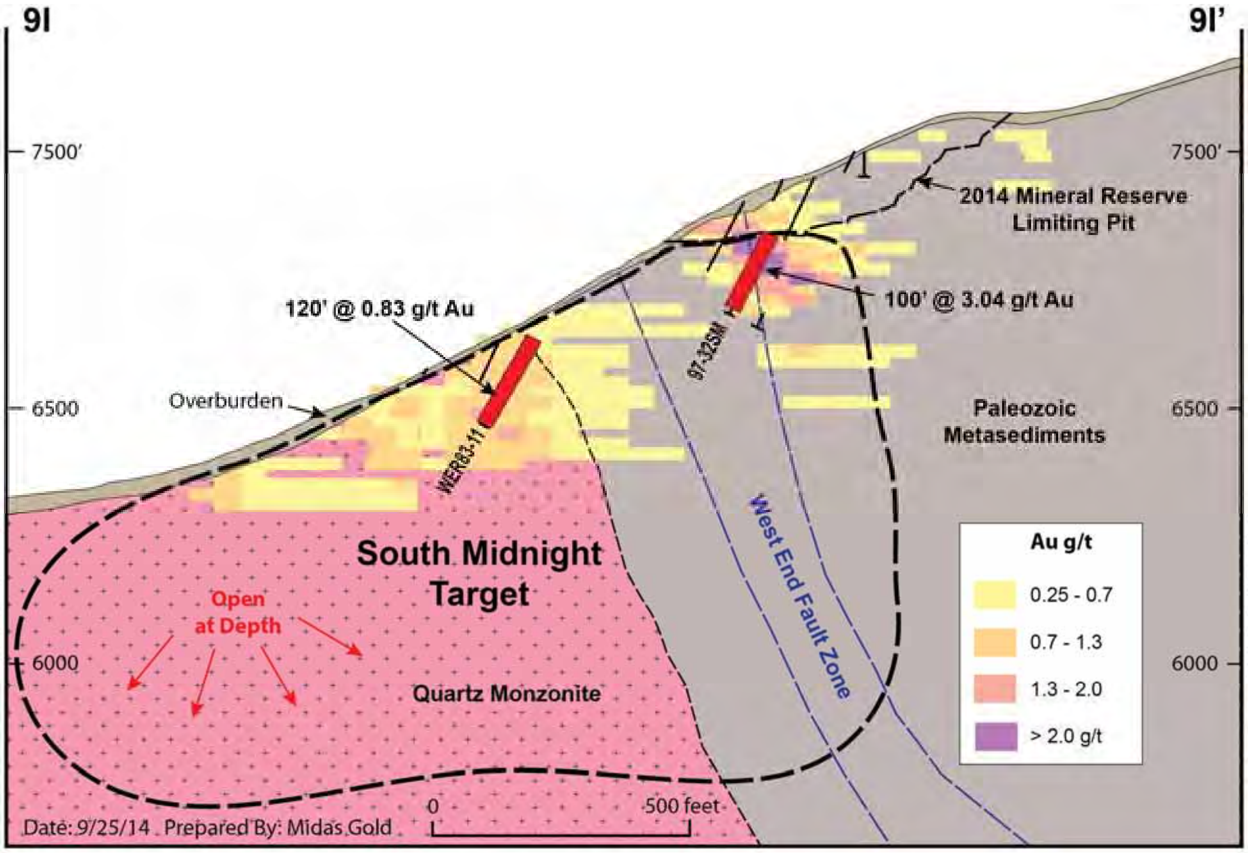


Note:
Potential mineralization reported here as a prospect may be partially included within our resources discussed in Section 14 of this Report.

South Midnight Target

This target is located at the southwest end of the West End Deposit. The area is identified by holes drilled by pre-Midas Gold operators. The most significant intercept was 100 ft averaging 3.04 g/t Au in hole 97-32SM (Figure 9.13). Mineralization roughly parallels the intrusive-metasediment contact and is open to the south.

Figure 9.13: Cross Section of the South Midnight Target with an 80 ft Corridor



Note:
Potential mineralization reported here as a prospect may be partially included within our resources discussed in Section 14 of this Report.

Exit Target

This target is located northwest of the main West End Fault Zone, but also includes an extension of the fault to the east-northeast. The area is identified by a strong surface soil and rock chip Au anomaly over an area of approximately 1,300 ft by 1,300 ft. Canadian Superior identified an apparently continuous zone of east-west trending mineralization over 360 ft wide averaging 0.72 g/t Au in chip samples from road cuts. Mapping and sampling by Midas Gold geologists confirmed portions of this anomaly where the road cuts were still accessible, the balance having been buried under waste rock storage facilities. This area has been inadequately tested by past drilling and warrants further study.

9.8 PROSPECTS FOR DISCOVERY OF HIGH GRADE, UNDERGROUND MINEABLE POTENTIAL

9.8.1 Scout

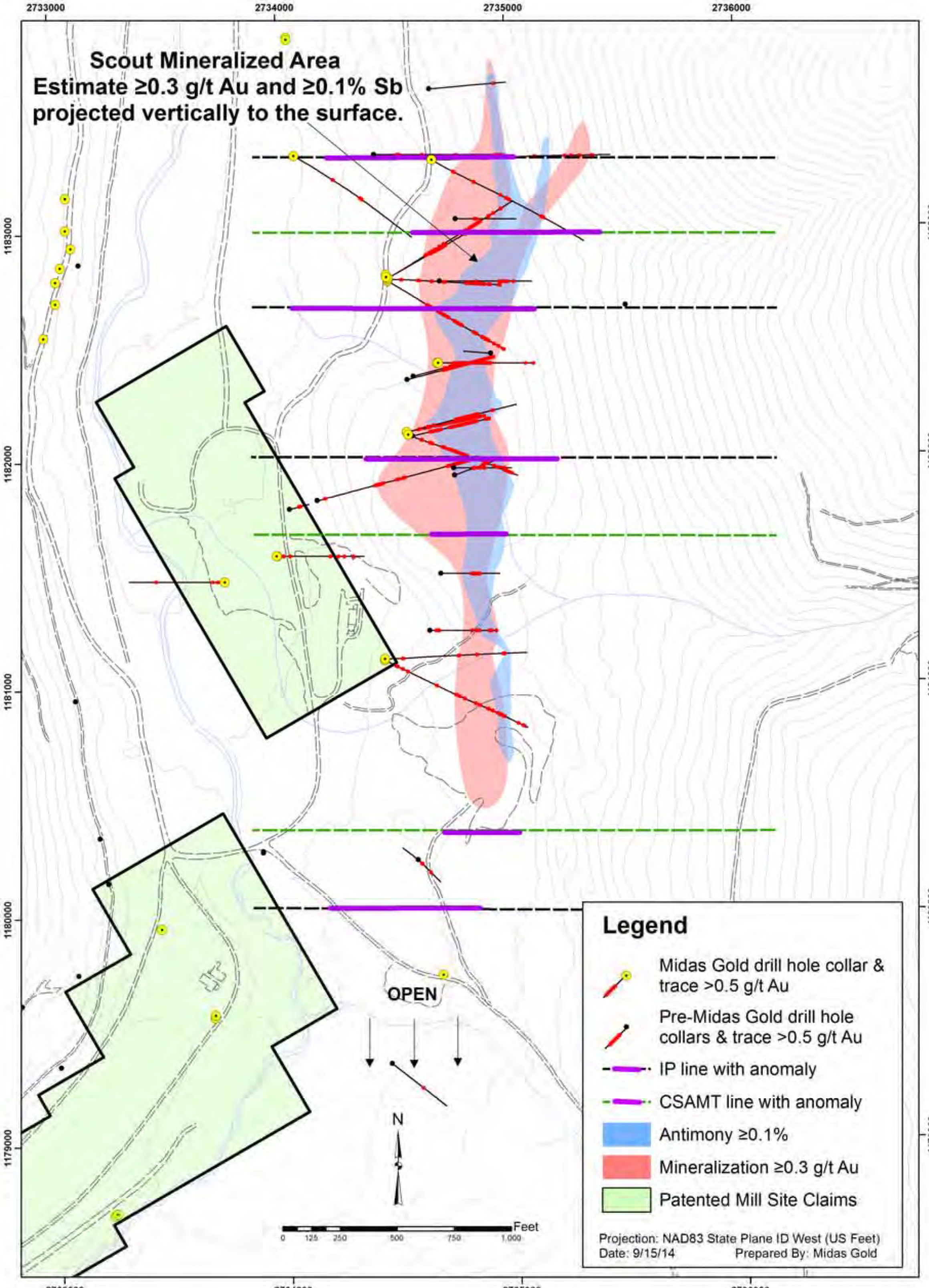
Scout is a potentially underground mineable Au-Ag-Sb exploration prospect (Figure 9.14) discovered in the 1930s by Bradley interests and further evaluated during Strategic Minerals investigations in the 1940s. Detailed exploration by other operators followed between 1947 and 1990. Pre-Midas Gold drilling includes 18 holes totaling 6,912 ft. Midas Gold work includes IP and CSAMT surveys, mapping, rock and stream sediment sampling along with the completion of 21 drill holes totaling 15,629 ft; Table 9.4 lists significant drill intercepts. Host rocks include quartzite, schist, quartz diorite and monzonite. Controls on mineralization are related to the Scout Valley Fault Zone, which trends north-south and dips steeply west, in addition to east-west and southwest-northeast trending faults. Drilling to date has been deemed insufficient to produce a mineral resource estimate, but does suggest a potential underground exploration target. The dimensions of the potential target, as determined by simple polygonal estimation methods from drilling, and further defined by trenches and geophysical surveys, outlines a conceptual potential underground target in the range of 2-5 million tons containing between 50,000-300,000 oz Au; 40,000,000-150,000,000 lbs Sb; and 300,000-1,500,000 oz Ag with target dimensions of approximately 25-75 ft thick (true), 2,000-3,000 ft along strike and extending 250-300 ft down dip at grades ranging from 1-2 g/t Au, 1-4% Sb, and 5-25 g/t Ag. Mineralization is open to the south, where monitoring well MWH-B08 cut 35 ft of 0.98 g/t Au and 40 ft of 0.97 g/t Au with 0.21% Sb coincident with an IP and CSAMT anomaly. Exploration data for the Scout target include geophysical data, geochemistry from soil, rock and trench samples, and results from widely spaced drill holes; as a result the potential size and tenor of the targets are conceptual in nature. There has been insufficient exploration to define mineral resources on these prospects and this data may not be indicative of the occurrence of a mineral deposit. Such results do not provide assurance that further work will establish sufficient grade, continuity, metallurgical characteristics and economic potential to be classed as a category of mineral resource.

Table 9.4: Selected Drill Intercepts from the Scout Prospect

Drill Hole ID	Operator	Collar Dip (˚)	Collar Azimuth (˚)	From (ft)	To (ft)	Interval (ft)	Au(1) (g/t)	Sb(1) (%)	Ag (g/t)
MGI-12-198-RC	Midas Gold	-90	-	294.9	335.0	40.0	0.83	1.9	-
				565.0	605.0	40.0	2.16	1.1	-
MGI-12-238	Midas Gold	-66	77	683.1	769.0	86.0	1.33	1.06	7.36
MGI-12-244	Midas Gold	-45	77	305.4	429.1	123.7	2.37	0.5	5.88
				including					
				331.4	363.8	32.5	5.7	1.46	15.3
MGI-12-249	Midas Gold	-53	115	311.7	862.5	550.9	0.78	2.02	14.8
				including					
				419.3	490.8	71.5	0.82	4.63	43.5
MGI-12-302	Midas Gold	-45	120	495.1	534.4	39.4	4.55	1.71	4.65
				651.6	677.5	25.9	1.68	2.86	8.42
MGI-12-345	Midas Gold	-44.6	116	764.1	816.9	52.8	1.68	5.42	48.0
MGI-12-347	Midas Gold	-50	90	771.3	784.4	13.1	5.96	12.3	114.6
MC-58	USBM	-20	75	625.0	730.0	105.0	1.77	0.3	-
MC-60	USBM	-45	77	109.9	419.9	310.0	1.0	0.33	-
				464.9	487.9	23.0	2.87	0.19	-
S-04-74	Superior	-45	90	276.6	325.8	49.2	-	1.44	-

Note:
(1) Selected intercepts composited with length weighted averages with cut-off grade of 0.75 g/t Au and/or 0.3% Sb reported, >10 ft composite length and <10 ft of internal waste below 0.5 g/t Au and/or 0.1% Sb. Reported drill intercepts are approximate true widths.

Figure 9.14: Plan map of the Scout Prospect



9.8.2 Garnet

The Garnet target is a potential underground mineable exploration prospect. It is the site of past underground exploration in the 1920s and a small open pit exploiting oxide heap leach ores in the mid-1990s. El Paso Oil and Gas discovered a broad zone of outcropping high-grade gold mineralization here in the mid-1970s and, between 1974 and 1995, four other companies explored the prospect. Pre-Midas Gold drilling includes 105 RC, core and air track percussion holes totaling 16,261 ft. The length-weighted average grade of pre-Midas Gold down-hole drill composites (using cutoff detailed in Table 9.5) is 5.3 g/t Au. Highlights of some of the drill intercepts in the unmined portions of the prospect are tabulated in Table 9.5. In 1995, Stibnite Mines, Inc. operated an open pit mine for one season within the prospect area. For approximate historic production records, see Section 6 of this Report. Fire assay grades of the material mined, where fire assayed, were approximately twice the cyanide leachable head grade. Mineralization occurs in sulfide and silica impregnated carbonates within a north plunging body developed at the intersection of a north-south striking, steeply to moderately west-dipping fault zone within two granite sills and an east-west striking, north-dipping dolomite unit. Midas Gold work includes mapping and rock, soil, and stream sediment sampling, but no drilling.

The conceptual target generally trends north-northwesterly from the Garnet Pit (Figure 9.15 and Figure 9.16). The dimensions of mineralized material located beneath and beyond the boundaries of the former open pit, as determined by simple polygonal estimation methods from historic drilling (Figure 9.15, Figure 9.16) and geophysical data, outlines a conceptual potential underground target in the 1-2 million ton range containing 250-500 koz Au approximately 30-60 ft thick (true) by 160-250 ft wide by 1,300-1,800 ft long down plunge at grades ranging from 5 g/t Au to 8 g/t Au. Exploration data for the Garnet target include geophysical data, geochemistry from soil, rock and trench samples, and results from widely spaced drill holes; as a result the potential size and tenor of the targets are conceptual in nature. There has been insufficient exploration to define mineral resources on these prospects and this data may not be indicative of the occurrence of a mineral deposit. Such results do not provide assurance that further work will establish sufficient grade, continuity, metallurgical characteristics and economic potential to be classed as a category of mineral resource.

Table 9.5: Pre-Midas Gold Drill Intercepts within the Garnet Prospect

Drill Hole ID	Operator	Collar Dip (˚)	Collar Azimuth (˚)	From (ft)	To (ft)	Interval (ft)	Gold(1) (g/t)
78-01GD	Superior	-90	0	175.0	190.0	15.0	18.52
G-07	Pioneer	-90	0	90.0	125.0	35.0	3.07
G-10	Pioneer	-90	0	100.0	185.0	85.0	3.06
RH76-25	El Paso	-90	0	143.0	158.0	15.0	9.13
S-23-76	El Paso	-55	337	186.0	196.5	10.5	7.18
S-29-76	El Paso	-62	189	215.0	287.0	72.0	3.35
				including			
				262.8	278.5	15.7	9.73
Xray-05-75	El Paso	-90	0	127.0	158.0	31.0	7.24
				including			
				135.0	148.5	13.5	15.56

Note:
(1) Drill hole composites over 3 g/t Au reported, >30 ft composite length and <10 ft of internal waste below 0.5 g/t Au. Higher-grade composites >6 g/t reported, >10 ft composite length and <5 ft of internal waste below 3 g/t Au. These intercepts are located beneath the bottom of the former open pit and estimated true widths are 80-90% of the reported intercept lengths.

Figure 9.15: Plan map of Grade x Thickness at the Garnet Prospect

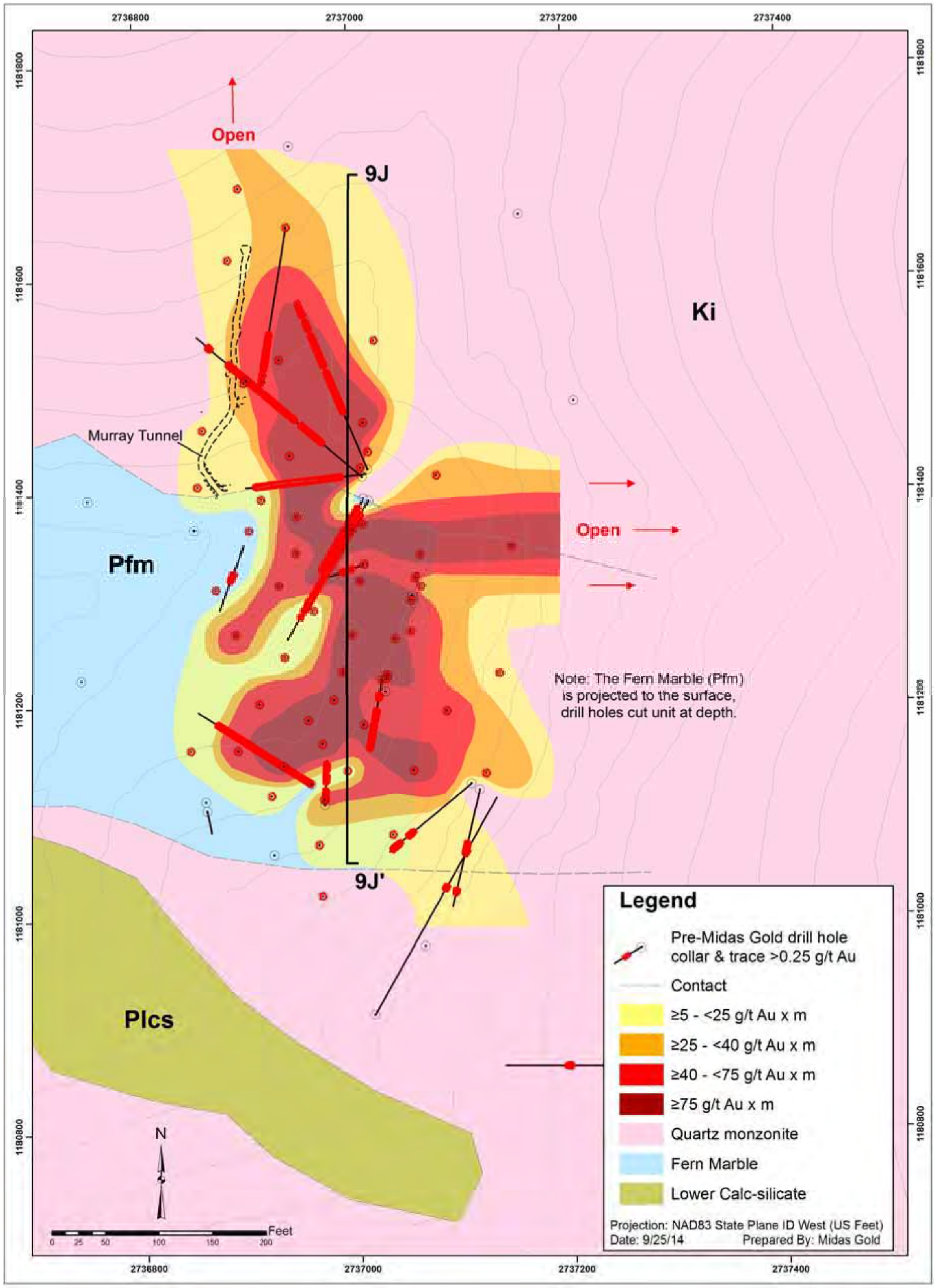
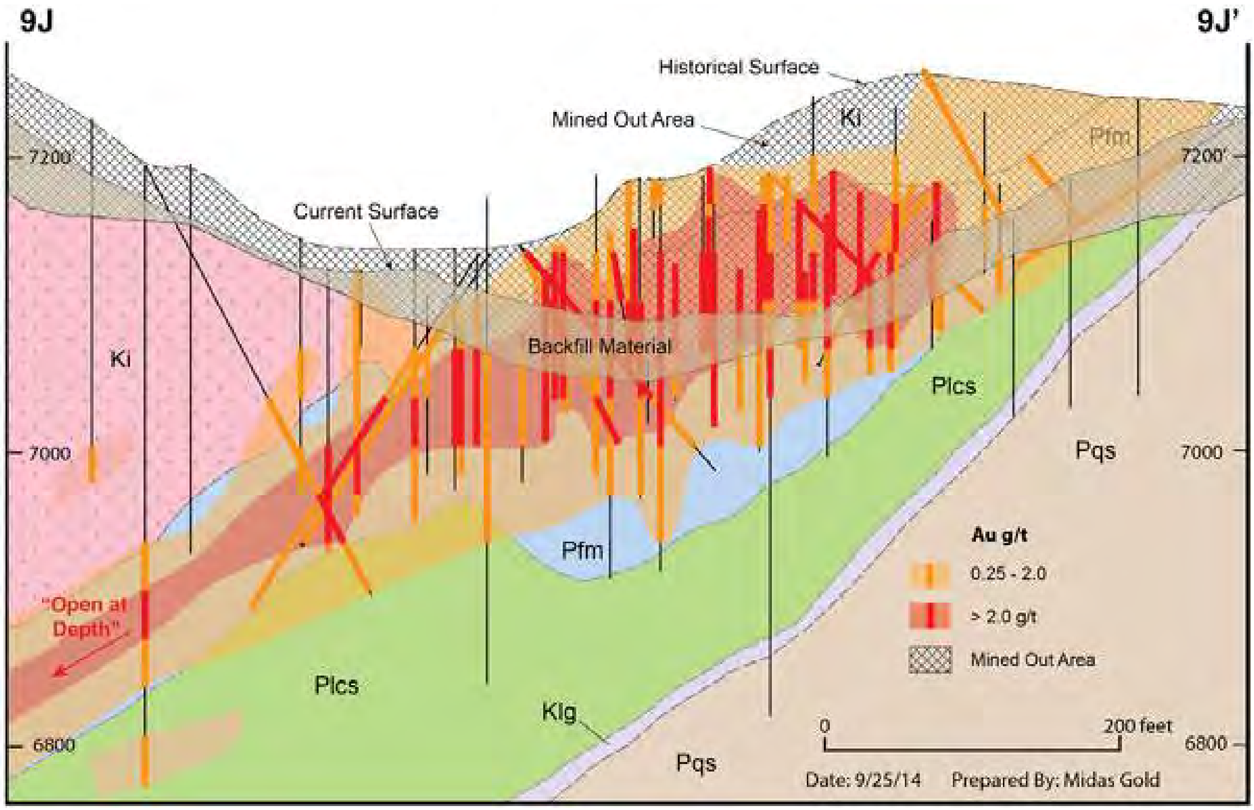


Figure 9.16: Cross Section of the Garnet Prospect with a 150 ft Corridor



9.8.3 Upper Midnight and Doris K

The Upper Midnight and Doris K prospects are located north-northeast and northeast of the Garnet Prospect, respectfully; they were originally located prior to World War II and re-discovered in the early 1970s when El Paso and Superior sampled and defined numerous large gold-in-soil and rock chip anomalies. In 1976, sampling of a black carbonate outcrop at Upper Midnight returned high grade gold assays, which were followed up by air track, core and RC drilling that confirmed the presence of a steeply southeast-dipping, northeast-striking high grade mineralized zone. Subsequent drilling campaigns included 2,349 ft in 28 shallow core, RC and air track percussion holes but did not adequately test the down dip extent or strike extensions of this zone, which appears to be approximately 60 ft thick (true width) with a length-weighted average grade of 8.33 g/t Au (Table 9.6). In the early 1990s, a large soil, ground magnetics and Very Low Frequency Electro-Magnetic (VLF-EM) survey was completed over both the Upper Midnight and Doris K prospects and outlined several coincident magnetic anomalies and strong conductive features spatially associated with the anomalous geochemical features. The Doris K prospect also saw significant soil, rock, and trench sampling, but has not been drill tested. Neither prospect has any recorded historic production.

Between 2010 and 2013, Midas Gold collected stream sediments, soils, and outcrop samples covering Upper Midnight and Doris K to confirm and expand on past exploration work. At Upper Midnight, 60 rock chip samples outlined anomalous gold values over a broad area including 3 ft chip samples of 5.24 g/t Au within brecciated quartzites and 2.79 g/t Au in altered carbonates within a large 400 ft by 500 ft soil anomaly (> 0.1 g/t Au). A total of 46 rock samples were collected at the Doris K prospect with 19 of the 46 samples resulting in values > 0.1 g/t Au. High values range up to a maximum of 15.7 g/t Au within the brecciated quartzite and 13.55 g/t Au within the altered carbonates. These higher grade samples were taken within 150 ft of each other and within a historical 300 ft by

250 ft gold-in-soils anomaly near a historical adit. Ground geophysical (CSAMT and IP-Resistivity) and airborne EM surveys produced anomalous responses over known or suspected mineralized zones in these areas and adjacent to them.

Mineralization at both prospects occurs in quartzites, carbonates and calc-silicates near the intersections with northeast and northwest striking faults. The prospects lie on the flanks of a large, northwest-trending, marble-cored, synform; Upper Midnight on the overturned northeast limb and Doris K along the hinge. The conceptual targets consist of sediment-hosted, structurally and stratigraphically controlled, high grade, potentially underground mineable gold deposits. The Upper Midnight target trends north-northeast, and lies at the intersection of faults and the Middle Marble unit. Due to steep slopes and small footprint, there is little to no open pit potential, but the mineralized zone, as defined by soils, rocks, trenches and drilling, could extend along strike for a strike length of 500-600 ft and down-dip to the northeast for 150-200 ft. With the grades encountered to date, Upper Midnight represents an excellent high grade potentially underground mineable target. Doris K is situated within a northeast trending zone of siliceous stockwork veining and breccias cutting quartzite and a friable recrystallized limestone unit. The breccia zone is approximately 50 ft-150 ft wide and is traceable for at least 500 ft in a northeast-southwest direction and is exposed over 200 ft of vertical extent across the nose of the synform. While less advanced than Upper Midnight, based on surface sampling to date, Doris K could represent another high grade, potentially underground mineable deposit.

Table 9.6: Pre-Midas Gold Drill Intercepts at the Upper Midnight Prospect

Drill Hole ID	Operator	Collar Dip (˚)	Collar Azimuth (˚)	From (ft)	To (ft)	Interval (ft)	Gold[(1)] (g/t)
EPE-78-01	El Paso	-90	-	0	112.0	112.0	5.56
				including			
				0	35.0	35.0	11.35
				100.0	112.0	12.0	15.91
PH-056	Superior	-90	-	0	50.0	50.0	4.62
PH-089	Superior	-90	-	0	100.0	100.0	6.73
				including			
				5.0	30.0	25.0	15.61
PH-094	Superior	-90	-	15.0	90.00	75.0	14.75

Note:
(1) Selected intercepts composited with length weighted averages with cut-off grade of 0.75 g/t Au and/or 0.3% Sb reported, >10 ft composite length and <10 ft of internal waste below 0.5 g/t Au and/or 0.1% Sb.

9.9 PROSPECTS FOR DISCOVERY OF NEW BULK MINEABLE POTENTIAL

9.9.1 Broken Hill - Saddle Trend

The Broken Hill, Ridgetop, Cinnamid, Saddle and Fern prospects are large tonnage, potentially bulk-mineable exploration targets that are located on a 2.5 mile long, northwest trending zone of continuous alteration and mineralization that also includes the Stibnite and West End deposits to the northwest and the former Fern Mine to the southeast. These prospects are shown in the East Side long section on Figure 9.2. Ridgetop and Cinnamid were discovered during soil sampling in 1990-1991; follow-up included systematic road cut rock chip sampling and mapping, with a total of 775 rock chip samples, followed up with drilling of 74 holes in Ridgetop and Cinnamid between 1992 and 1996. Drill hole 92-47 intersected 74.8 ft grading 2.13 g/t Au at Ridgetop and drill hole 92-49 intersected 52.5 ft averaging 4.15 g/t Au at Cinnamid. Highlights of historical drilling are tabulated in Table 9.7. Broken Hill was discovered by rock chip sampling in 1991 and followed with limited trenching and drilling. The Saddle prospect was discovered during soil sampling in the early 1980s and follow up rock chip sampling which outlined a broad area, approximately 1,500 ft in length by up to 500 ft in width, of anomalous gold in silicified Fern

Marble. Based on surface rock-chip and soil sampling between the Cinnamid and Saddle prospects, there appears to be continuity of gold mineralization at surface, in the area separating the two prospects over a distance of about 2,000 ft. A water monitor well drilled in 1996 confirmed the strong gold-in-soils anomaly at Saddle, intersecting 120 ft of 0.92 g/t Au in oxidized Fern Marble (Table 9.7). Since most analytical work from drill sample assays along the trend utilized CN-soluble assay methods, total gold grades may be significantly under reported. Midas Gold has mapped and rock-sampled all of the prospects along this trend, but has not completed any additional drilling. In 2013, Midas Gold expanded the soil grid northeast of Broken Hill, which generated a large soil anomaly on strike with the northeast structures controlling mineralization at Broken Hill, suggesting these structures have additional potential along strike where no past work has been completed.

Geologically, the Broken Hill - Saddle Trend is defined by the intersection of northwest-striking, northeast dipping metasediments and district-scale northeast-striking moderate to high angle faults that provided the conduits for gold mineralization. Gold was preferentially deposited in reactive siltites, calc-silicates, Fern Marble and in a sequence of interbedded quartzites and schists) along the trend. Ridgetop and Cinnamid have been drilled extensively by previous operators, who focused on shallow oxide mineralization, while a nominal number of holes have been drilled at Broken Hill and only a single water monitoring well was drilled at Saddle. However, excellent potential exists along the entire trend to discover additional sulfide mineralization. On the southeast end of the trend, between the Ridgetop and Saddle prospects, drilling, trenching and rock chip sampling has defined a body of mineralized material occurring as stacked lenses within the lower Calc-silicate, Fern Marble and quartz-schists sequences with an aggregate true thickness ranging from between 75-125 ft (200-325 ft in plan view) that is from 3,000-3,500 ft along strike and extends approximately 200-325 ft vertically below the ground surface (the limits of current drilling), defining a conceptual target ranging from 4-10 million tons at grades between 1-2 g/t Au. All previously drilled mineralization remains open to expansion in along strike and down dip. Given the trend is over 2.5 miles long and is only drill tested over a short section of the trend, there is considerable upside potential. Exploration data for the target area include geophysical data, geochemistry from soil, rock and trench samples, and results from widely spaced drill holes; as a result the potential size and tenor of the targets are conceptual in nature. There has been insufficient exploration to define mineral resources on these prospects and this data may not be indicative of the occurrence of a mineral deposit. Such results do not provide assurance that further work will establish sufficient grade, continuity, metallurgical characteristics and economic potential to be classed as a category of mineral resource.

The Fern Prospect is an epithermal, carbonate-hosted, Au-Ag exploration target at the southeast end of the Broken Hill-Ridgetop-Cinnamid-Saddle trend that was initially explored for gold in the early 1900s and later was prospected for mercury during the Thunder Mountain gold rush (Larsen and Livingston, 1920). There was minor Hg production after the turn of the century from ores developed from open cuts and from underground workings over a total elevation of approximately 1,085 ft. In the 1940s, the USBM conducted mercury exploration, including extensive trenching. In the 1950s, additional trenching was completed under the DMEA program and, in the early 1960s, additional sampling, trenching and drilling of 5 holes totaling 1,503 ft were completed under an Office of Mineral Exploration (OME) contract. In 1983-1984, rock chip, soil sampling, detailed mapping and two geophysical IP lines were completed by Canadian Superior Mining, who drilled three follow-up RC holes totaling 1,780 ft in 1984. Pioneer did some follow up rock-chip sampling in 1987 and drilled five holes totaling 2,400 ft in 1990. Further soil sampling was completed by Barrier Reef Inc. in 1990. There is potential to discover a low-tonnage, high-grade, potentially underground mineable gold deposit and/or mineralization amenable to open pit extraction methods where low-grade gold mineralization would be associated with replacement bodies in the Fern Marble. Twenty-nine systematic outcrop chip samples, taken by Pioneer Metals in 1987 (near an old adit driven adjacent to a northeast trending jasperoid breccia) ranged from 0.87 g/t Au to 42.7 g/t Au and averaged 14.3 g/t Au. Holes drilled nearby at the time did not adequately test the mineralized structure but, nevertheless, the holes did cut significant mineralization (Table 9.7), suggesting the possibility of a larger, open-pit target, as well as underground potential from the main structure. Midas Gold work has included mapping, rock, soil and stream sediment sampling. Midas Gold collected a total of 11 systematic chip samples that averaged 19 g/t Au across a northeast trending, northwest dipping, 25-65 ft wide breccia. Mapping traced the zone of mineralization for approximately 175 ft along strike and over 130 ft of vertical relief up slope, beyond which it is lost under cover in both directions.

Table 9.7: Significant Drill Intercepts within the Broken Hill – Saddle Trend

Prospect	Operator	Drill Hole ID	Collar Dip (˚)	Collar Azimuth (˚)	From (ft)	To (ft)	Interval (ft)	Gold(1) (g/t)
Broken Hill	Pioneer	91-31	-90	N/A	30	110	80	1.06
Ridgetop	Pioneer	92-47	-90	N/A	135	205	70	2.23
Ridgetop	SMI	95-63	-90	N/A	155	190	35	2.06
Ridgetop	SMI	96-67	-90	N/A	105	180	75	2.41
Cinnamid	Pioneer	92-49	-90	N/A	235	285	50	4.54
Cinnamid	SMI	95-69	-90	N/A	15	90	75	2.56
Cinnamid	SMI	95-70	-90	N/A	110	185	75	3.06
Cinnamid	SMI	96-62	-90	N/A	330	460	130	1.23
Saddle	SMI	MW-96-01	-90	N/A	25	145	120	0.92(2)
Fern	Pioneer	90-36	-55	215	205	235	30	2.73
					345	365	20	1.36
Fern	Pioneer	90-34	-50	222	285	300	15	1.01
Fern	Pioneer	90-37	-60	180	85	120	35	0.77

Notes:
(1) Selected intercepts composited with length-weighted averages of continuous mineralization with over 0.75 g/t Au reported, >10 ft composite length and <10 ft of internal waste below 0.5 g/t Au.
(2) Cyanide assay method only.

9.9.2 Hermes Trend

Similar to the Broken Hill-Saddle trend, the Hermes Trend prospects are located along a northwest-southeast zone that follows the trace of the Hermes Marble unit over a distance of 2.5 miles from the West End Deposit to the Cinnabar Mine complex; the trend includes the Photon, AC/DC, and Hermes prospects.

On the northwestern end of the trend, the "Northeast Extension" portion of the West End Deposit lies within highly silicified Hermes Marble, where a series of northeast trending faults cuts the reactive and receptive unit. The Photon Prospect is a new discovery made by Midas Gold in 2013 that is comprised of a large, multi-element soil anomaly at the intersection and extension of the Broken Hill fault system and the Hermes Marble. Farther to the southeast, at the AC/DC Prospect, Midas Gold has developed a large soil anomaly and identified alteration in quartzites where anomalous rock chips in the area suggest a similar setting to the Photon Prospect. At the far southeast end of the trend, at the Hermes Prospect, a limited drilling program was completed in 1990 following up on a soil and rock chip anomaly where drill intercepts included 45 ft of 0.74 g/t Au, 15 ft of 0.7 g/t Au, 40 ft of 0.45 g/t Au, and 20 ft of 0.63 g/t Au. Many of the holes were only assayed utilizing CN-soluble methods and the lack of fire assays below the redox boundary may have significantly under reported total gold values for these areas. Other exploratory work along the Hermes trend includes extensive development work and drilling at the past producing Cinnabar Mine complex, which was operated intermittently between 1921 and 1958. During World War II, it was among the leading quicksilver (Hg) producers in the U.S. and was the second largest mercury producer in Idaho. Midas Gold work includes mapping, rock and soil sampling and limited ground geophysical surveys along the northwestern end of the trend.

Along the Hermes trend, silicification in the marble is pervasive, especially adjacent to quartzite contacts, and mineralization occurs in stratiform silicified zones where reactive carbonates have been converted to jasperoids, as well as in cross-cutting breccias and silica veined zones. The mineralized areas are typically elongated parallel to bedding strike, but are thickest and best developed at the intersection of favorable beds and northeast-trending fault structures, as seen at Photon. The current surface area of the soil and rock chip geochemical anomaly measures approximately 1,500 ft long and is up to 600 ft wide.

9.9.3 Mule

The Mule prospect is a potentially open pit and/or underground gold prospect associated with high grade sulfidic quartz veins and lower grade disseminated mineralization hosted in intrusive rocks. The prospect lies adjacent to the contact of the Stibnite roof pendant and volcanic stratigraphy associated with the Tertiary Thunder Mountain Caldera and can be seen on the East Side long section on Figure 9.2. Old surface cuts and minor underground workings from the 1920s era were rediscovered in the late 1980s. Subsequently, Pioneer excavated three trenches in 1987 around the northern end of some of the open cuts. These trenches cut a vein system that trends N30ºE and dips 30ºW. The first trench included a 1-2 ft wide vein which averaged 51 g/t CN-leachable Au within a 40 ft wide altered zone that averaged 0.58 g/t CN-leachable Au (excluding the vein intercept). The second trench, located approximately 175 ft North of the first, included a 2 ft wide vein which assayed 6.03 g/t CN-leachable Au within a 158 ft wide zone (true width ~75% of this) that averaged 0.4 g/t CN-leachable Au (excluding the vein). A third trench, 100 ft south of the first trench, did not intersect the vein nor cut altered rocks and no assays were reported, but appears to be situated too far to the west to intercept the vein based on the projections from the northern trenches. Midas Gold's 2011 airborne magnetic and EM surveys outlined a large, several mile long, N-S trending, geophysical feature running through this area and continuing to the north, through the Fern and Cinnabar mines. This survey resulted in geophysical characteristics similar to the Meadow Creek Fault Zone farther west that hosts both the Yellow Pine and Hangar Flats deposits. Follow-up work included mapping and rock, soil, and stream sediment sampling. In early 2012, a 550-sample reconnaissance type soil grid was established over the area and outlined two large soil anomalies near the old trenches and another anomaly farther to the south. A total of 18 rock samples, from the limited bedrock exposures within and around these soil anomalies, were collected from the prospects and consistently indicated the presence of narrow, but high grades of gold in veins within broader zones of silicified intrusive rocks.

Unlike mineralization elsewhere in the District, neither arsenic nor antimony appear to be particularly anomalous, nor associated with gold mineralization in this prospect. Sampling by Midas Gold outlined two large soil and rock chip anomalies associated with sericitized and silicified granite and high grade Au-veins and silicified Au-bearing intrusives. The largest of the anomalies in the south, is at least 1,500 ft long in a N-S direction and 750 ft wide in an E-W direction and also continues to be open to the south. The northern anomaly covers an area of approximately 350 ft by 750 ft, which is associated with the area of past historic trenching abuts against transported cover areas. The narrow, but high grades of gold in veins and silicified zones in the intrusive rocks could represent possible underground exploration targets while the historic trench results and large soil anomalies suggest two broad areas that might host potentially bulk tonnage mineralization amenable to open pit mining.

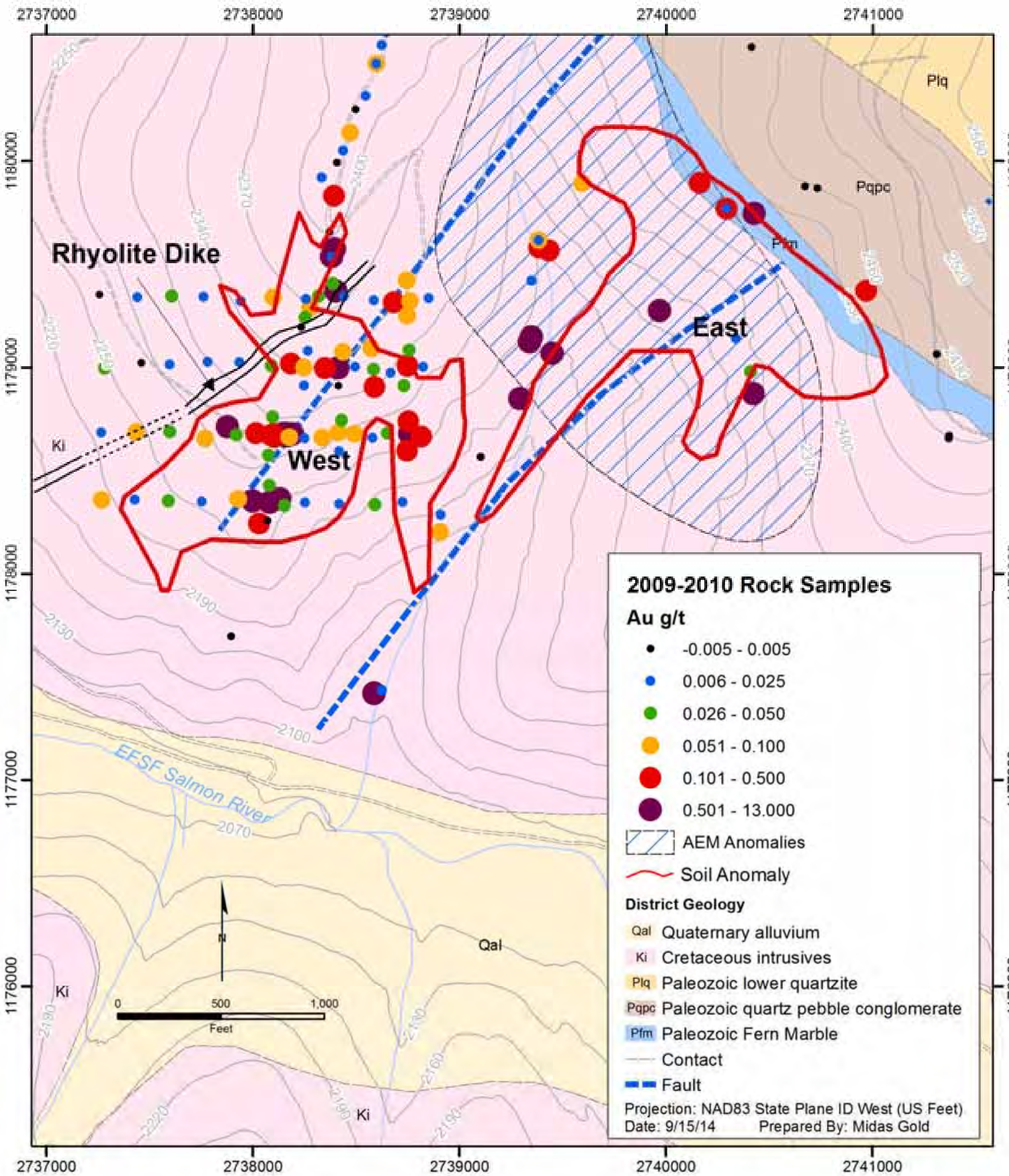
9.9.4 Rabbit

The Rabbit prospect consists of large, coincident, multi-lobed soil, rock chip geophysical anomalies situated east of the Hanger Flats Deposit and southeast of, and along strike from, the Garnet Prospect. The area was targeted after compilations suggested a setting similar to the nearby Garnet Prospect. Figure 9.17 shows soil and rock chip values and the two targets identified on the prospect. Mineralization occurs in both areas and in the intervening area, and is associated with silica, clay and sulfide impregnations and with extensive quartz-sulfide veining. Textural features suggest an epithermal environment. Midas Gold work included mapping, stream sediment, rock chip, soil and test pit sampling and two lines of IP-resistivity.

The intrusive-hosted West Rabbit Prospect was first discovered in the 1920s when a short (~250 ft long) adit was driven into altered quartz monzonite and minor placer workings were excavated in the creek below the adit. The

NE-trending lode is reported to be over 100 ft wide and to extend over a vertical range of 100 ft. Minor prospecting was completed by modern explorers, but the prospect has reportedly never been drilled. The West Rabbit area is underlain by anomalous soils and rocks (Figure 9.17) that outline a conceptual intrusive-hosted target zone roughly 825 ft wide x 1,475 ft long, with over 500 ft of vertical relief. The dolomite-hosted East Rabbit Prospect was discovered by Midas Gold geologists in 2010 (Figure 9.17) based on anomalous soils and rocks (Figure 9.17) that outline a conceptual target zone roughly 650 ft wide x 1,975 ft long, and with over 600 ft of vertical relief.

Figure 9.17: Plan map of the Soil Sample Geochemistry at the Rabbit Prospect



9.9.5 Meadow Creek Fault Zone Trend

The MCFZ trend consists of a ~2 mile long string of prospects aligned along the MCFZ and associated cross structures; at the southern end lies the Hangar Flats Deposit and at the northern end, the Yellow Pine Deposit. The prospects along this trend can be seen in the West Side long section on Figure 9.2. Targets vary from open-pit to underground exploration prospects. The North Tunnel, sunk in the Fiddle Creek drainage in the 1920s, was a short exploration tunnel, with minor production, that was later re-opened in the 1940s to complete a small underground drilling program. The DMEA tunnels were driven westward, towards the MCFZ, between the North Tunnel and the Meadow Creek Mine workings and discovered high-grade mineralization during underground sampling and drilling, but it was not exploited. Minor additional historic exploration occurred along the MCFZ Trend from Yellow Pine to Hangar Flats, including ground-based geophysical surveys, soil grids, trenches, pits and rock sampling.

The MCFZ Trend hosts mineralization in both high-grade, Au-Sb-Ag-W vein systems and disseminated intrusive Au-Sb-Ag mineralization. The Idaho batholith is the predominant rock unit along the trend but some metasedimentary rocks may be present, as suggested by drill intercepts and geophysical indicators. The majority of the trend is covered with glacial outwash deposits and there has been only limited drilling along the trend. Evidence of mineralization is mostly derived from previous underground exploration workings and limited widely spaced surface and underground drilling. The main MCFZ has been mapped underground as north-south trending and steeply dipping, with moderately-dipping, northeast and east-west striking structures intersecting it. Pre-Midas Gold underground mapping at the DMEA, Monday and North tunnels outlined extensive zones of Au-Sb-W mineralization and demonstrates potential for high-grade mineralization along the trend. Beneath the Fiddle Creek drainage, in the Monday Tunnel, an intercept of 240 ft grading 1.1% Sb and 0.7 g/t Au was reported just east of the main MCFZ trend. Midas Gold drilling in this area resulted in an intercept 45 ft of 2.02 g/t. In the DMEA workings north of the Hangar Flats Deposit, intercepts of Au-Sb-W mineralization are common in northeast trending shear zones and disseminated within intrusive rocks. Representative intercepts, when taken in aggregate (Table 9.8) are not considered underground grade at this time but high grade vein systems within these intervals with values >5 g/t Au along with significant W, Sb and Ag values, indicate the potential for high grade discoveries Figure 9.2. Broad soil and ground geophysical anomalies covering the projected surface expression of these vein and shear systems in the North DMEA area, and along the trend, suggest continuity of mineralization from these underground zones up to the surface.

From 2009 through 2012, Midas Gold completed over 31 line-mi of ground geophysical surveys, including IP-Resistivity and CSAMT, along with soil, rock and trench sampling and drilling in the DMEA and Fiddle Creek areas (Table 9.8). Additional work included a stream-sediment sampling program and several soil grids along the MCFZ trend. Drill intercepts were encouraging, especially in light of the fact that they did not drill into the main portion of the IP anomaly.

Table 9.8: Drill Intercepts within the Meadow Creek Fault Trend

Drill Hole ID	Operator	Collar Dip (˚)	Collar Azimuth (˚)	From (ft)	To (ft)	Interval (ft)	Gold(1) (g/t)
FC-1	USBM	+1	118	15	65	50	0.60
				290	345	55	0.92
MGI-10-39	Midas Gold	-45	83	245	290	45	2.02
DMA-05	USBM	+3	260	15	70	55	0.62
DMA-06	USBM	-1	157	35	70	35	0.71
				315	345	30	0.46
DMA-07	USBM	0	89	5	80	75	0.93
				235	270	35	0.51
DMA-08	USBM	0	270	10	100	90	0.78
DMA-09	USBM	0	270	105	170	65	0.50
DMA-10	USBM	0	106	0	55	55	0.76

Note:
(1) Drill hole composites over 0.5 g/t Au reported, >30 ft composite length and <15 ft of internal waste below 0.5 g/t Au.

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10 DRILLING

10.1 INTRODUCTION

The District has been drilled by numerous operators over the past 85 years. Table 10.1 below shows the number of holes and footage catalogued within the Midas Gold database consisting of a variety of drilling types including: percussion, auger, churn, core, reverse circulation (RC), rotary, and sonic drilled from both underground and surface drill stations.

Table 10.1: Pre-Midas Gold and Midas Gold Drilling by Mineralized Area

Mineralized Area	Pre-Midas Gold Drilling		Midas Gold Drilling		Total Drilling	
	# Holes	Feet	# Holes	Feet	# Holes	Feet
Yellow Pine	768	147,936	242	152,009	1,010	299,945
Hangar Flats	116	30,377	141	105,708	257	136,085
West End	889	208,260	55	38,907	944	247,167
Historic Tailings	25	1,512	47	3,786	72	5,298
Scout	18	6,912	21	15,629	39	22,541
Other	240	52,472	44	10,235	284	62,707
Totals	2,056	447,469	550	326,275	2,606	773,744

Notes:
(1) For clarity the numbers in the table have been rounded to the nearest whole number.

Pre-Midas Gold drilling was completed in conjunction with several surface and underground mining operations. Midas Gold drilling has been conducted for the purposes of exploration, mineral resource confirmation and definition, metallurgy, geotechnical engineering and condemnation drilling. The location of each mineralized area along with their associated drill hole collars can be found on Figure 10.1.

The Yellow Pine mineralized area has been drilled by 10 operators over the past 75 years and the total Yellow Pine database comprises approximately 299,945 ft of drilling in 1,010 holes. Drilling employed a variety of methods including core, RC, rotary, and air track (Table 10.1 and Figure 10.2). The pre-Midas Gold drilling was primarily performed in conjunction with surface and underground mining operations.

The Hangar Flats mineralized area has been drilled by six operators over the past 85 years totaling approximately 136,085 ft of drilling in 257 holes (Table 10.1 and Figure 10.3). Drilling employed a variety of methods including surface and underground core, RC, rotary, and sonic. The pre-Midas Gold drilling was primarily performed in conjunction with underground mining operations.

The West End mineralized area has been drilled by six operators over the past 74 years and the total West End database comprises approximately 247,167 ft of drilling in 944 holes (Table 10.1 and Figure 10.4). Drilling employed a variety of methods including core, RC, rotary, and air track. The pre-Midas Gold drilling was primarily performed in conjunction with surface mining operations.

The Historic Tailings area has been drilled by 2 operators over the past 20 years and the total Historic Tailings database comprises approximately 5,298 ft of drilling in 72 holes. Drilling employed a variety of methods including RC, sonic, and auger (Table 10.1 and Figure 10.5). Pre-Midas Gold drilling was conducted for well construction.

The Scout prospect has been drilled by 5 operators over the past 60 years and the total Scout database comprises approximately 22,541 ft of drilling in 39 holes. Drilling employed a variety of methods including core, RC, and air track (Table 10.1 and Figure 10.6). All drilling at Scout has been conducted as exploration drilling.

Figure 10.1: Mineralized Areas Showing All Drill Hole Collars

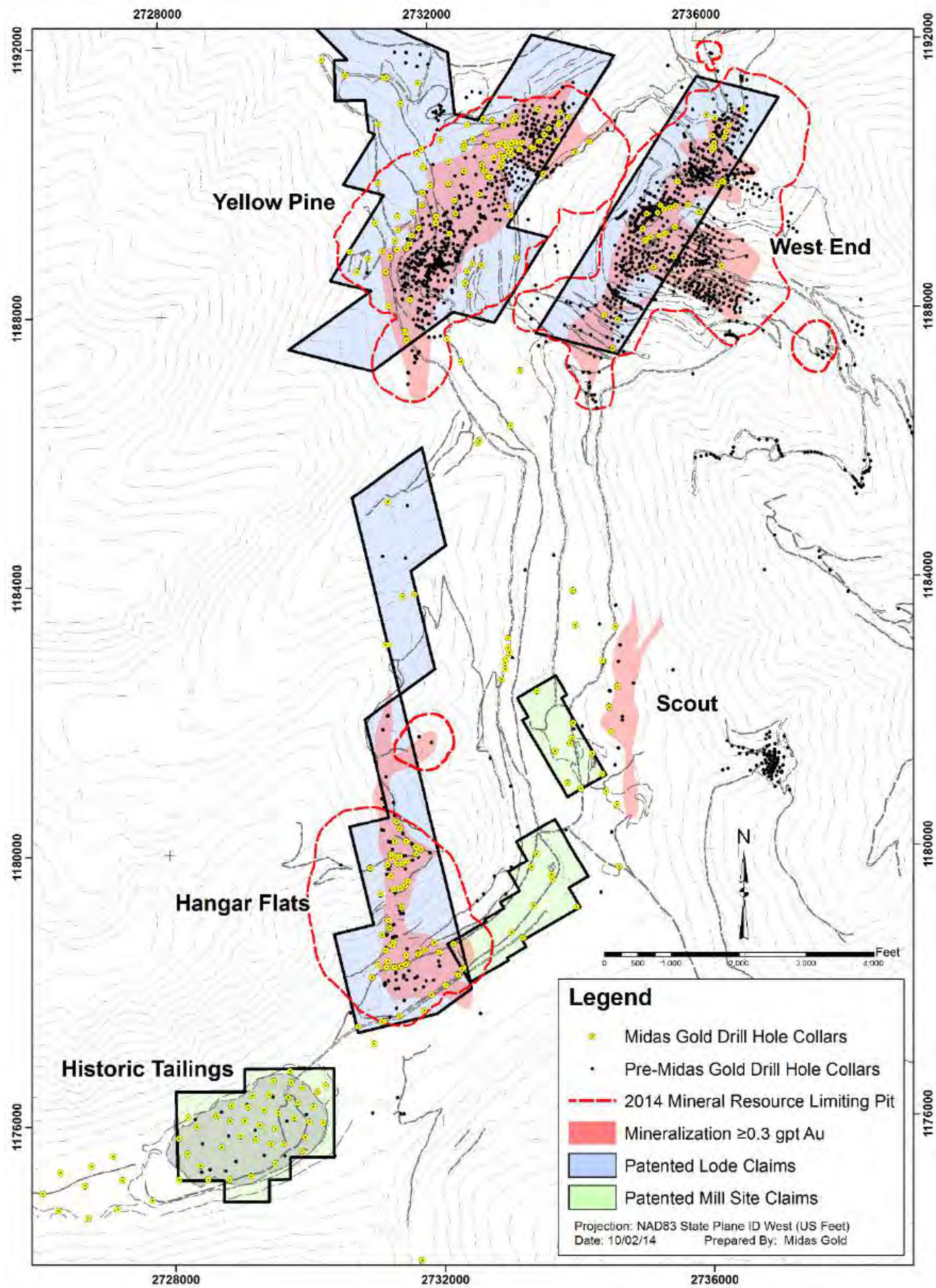


Figure 10.2: Yellow Pine Mineralized Area Showing All Drill Holes

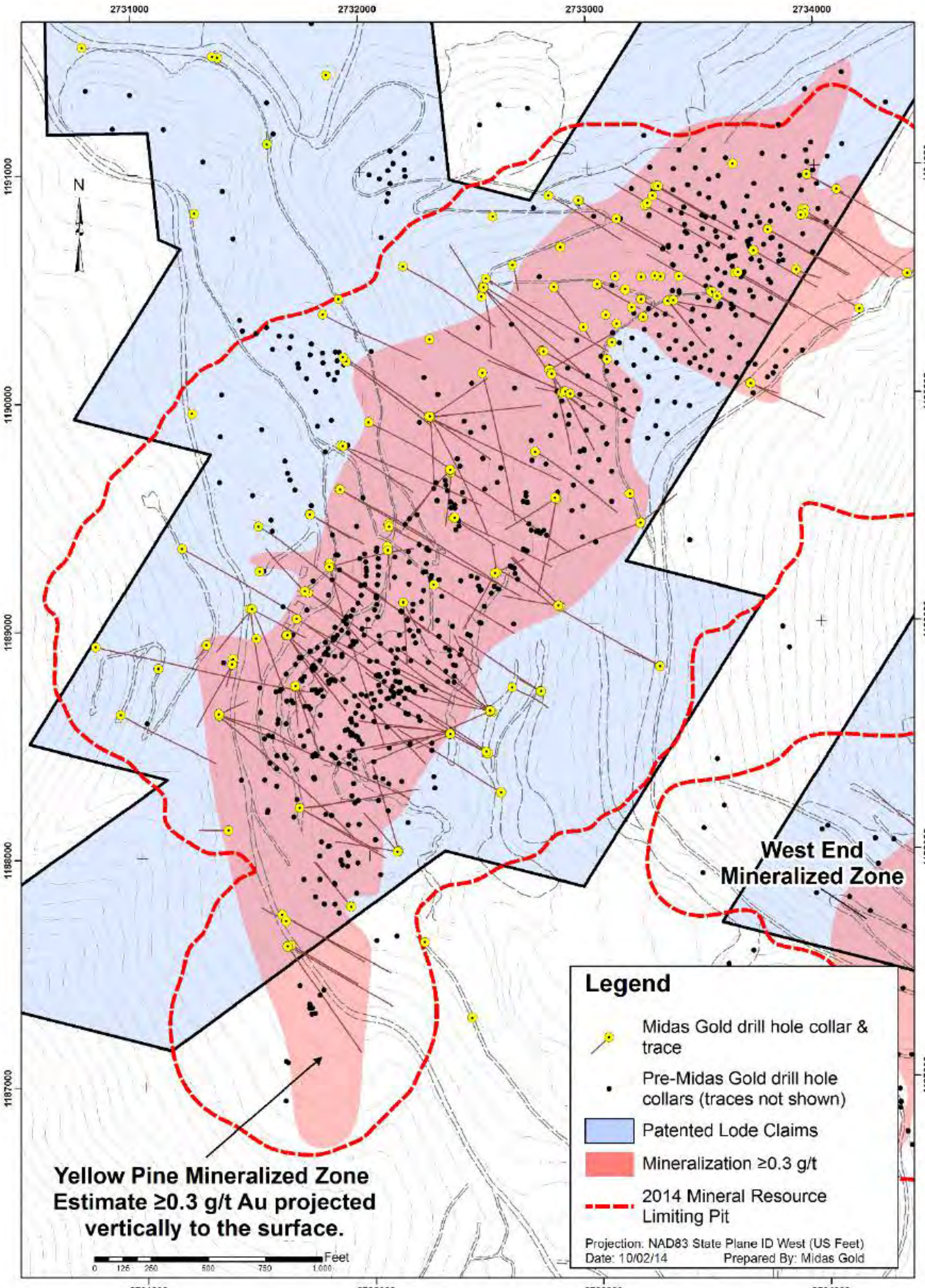


Figure 10.3: Hangar Flats Mineralized Area Showing All Drill Holes

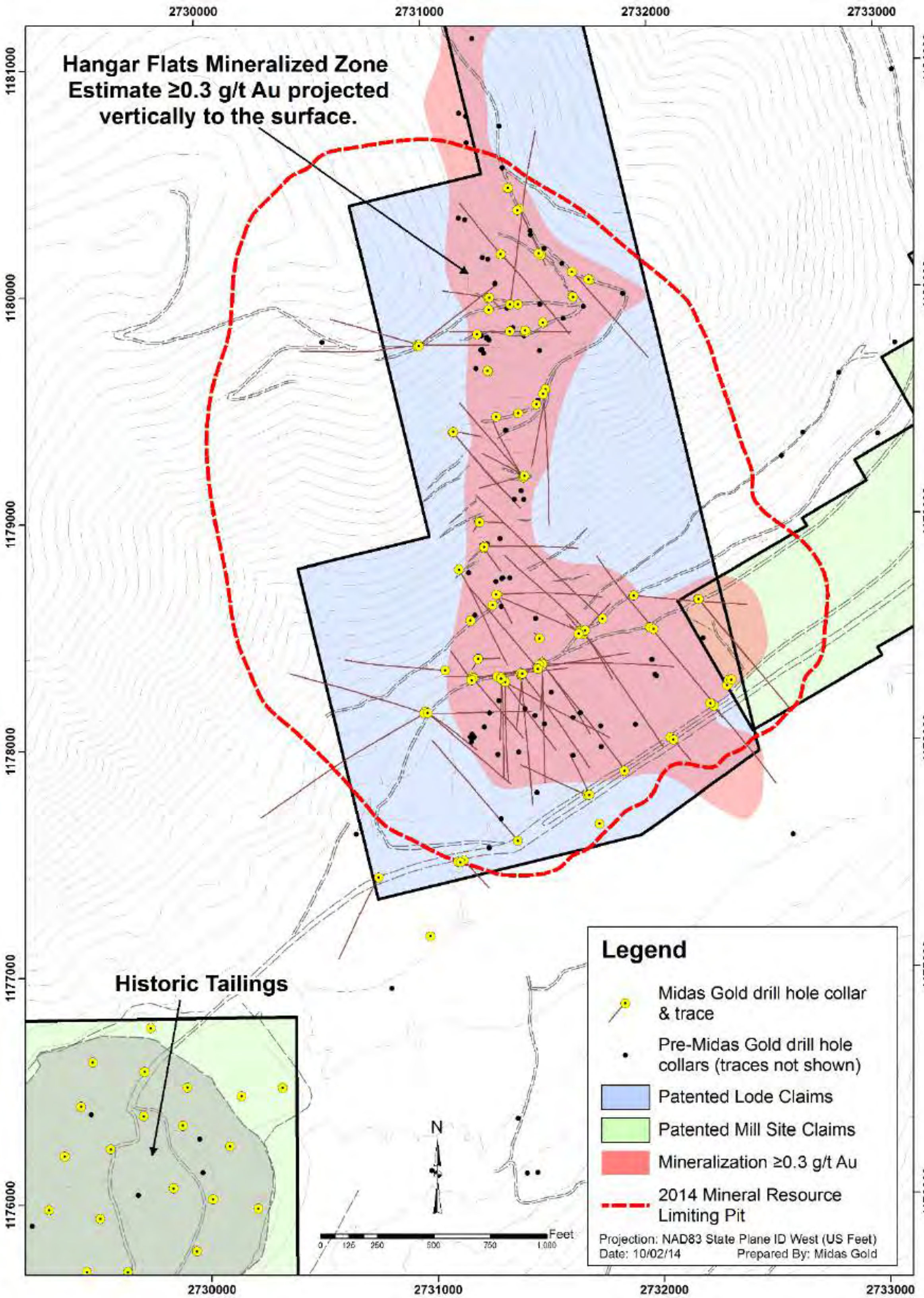


Figure 10.4: West End Mineralized Area Showing All Drill Holes

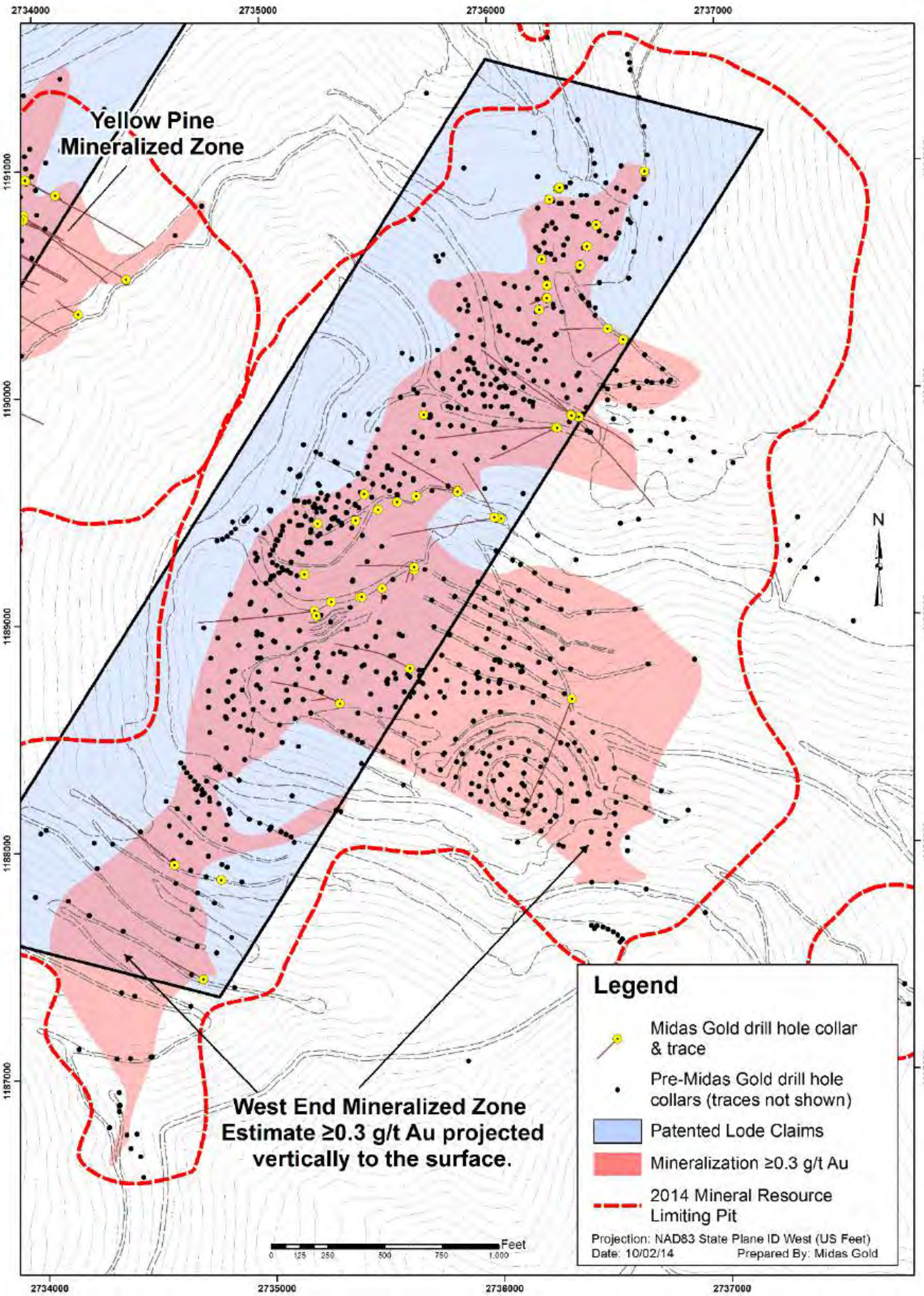


Figure 10.5: Historic Tailings Area Showing All Drill Holes

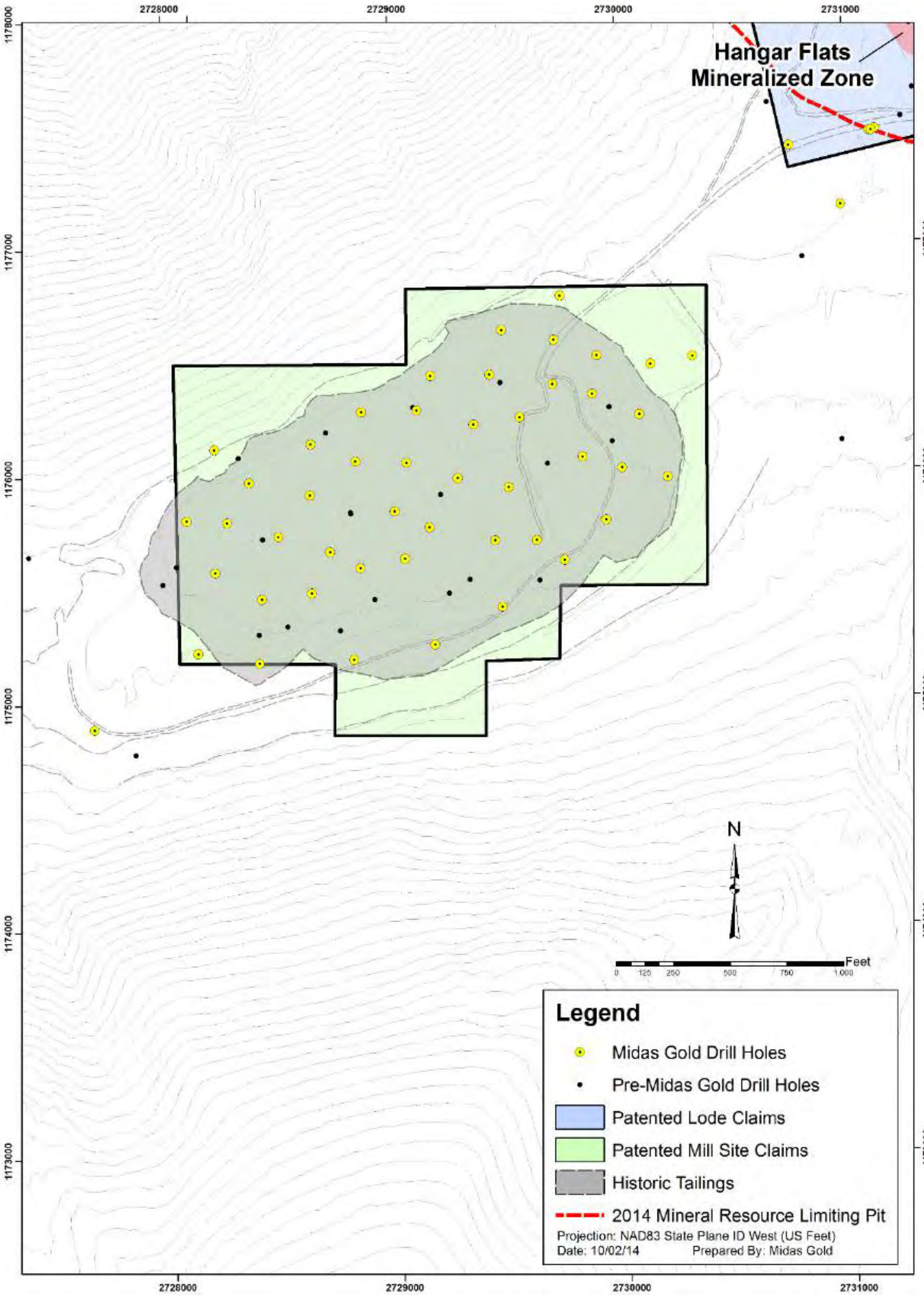
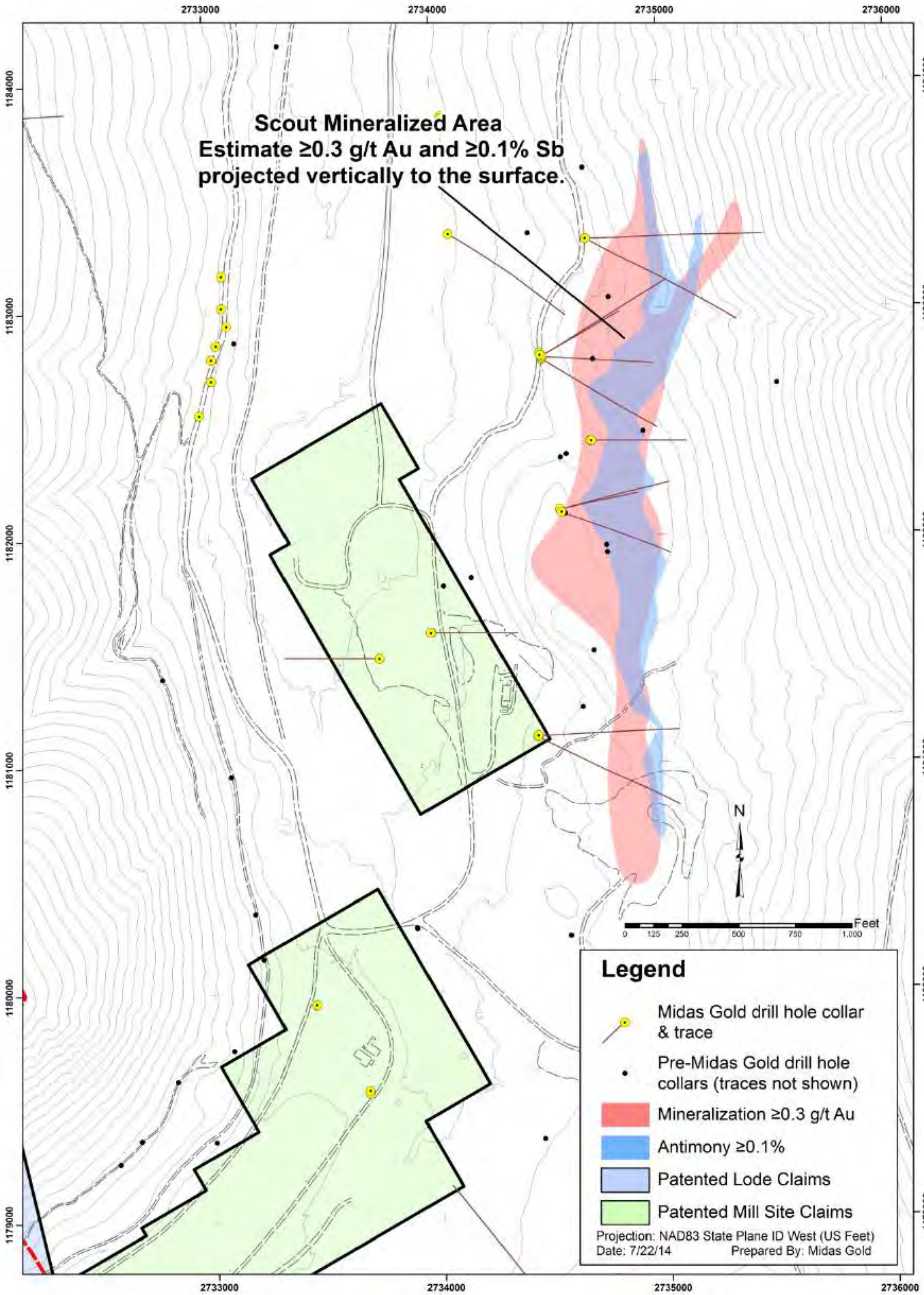


Figure 10.6: Scout Prospect Showing All Drill Holes



Drill holes in the mineralized areas were drilled on a variety of orientations to intersect north-, northeast-, and northwest- striking structural features which control mineralization. Twenty-nine percent of pre-Midas Gold holes were drilled vertically, 46% were drilled on a westerly to north westerly azimuth, the remaining were drilled on a southwest or southeast or easterly azimuth. Midas Gold holes were drilled on several different azimuths; 31% were vertical holes, 26% were drilled on a southeast azimuth, 19% on a northwest azimuth, 14% were drilled easterly, and 10% on a southerly azimuth.

10.2 DRILLING METHODS

Many drilling methods have been used by previous operators and by Midas Gold. Methods have varied by operator, time period, and deposit across the District. Methods have included air track, auger, churn, both surface and underground core, RC, rotary, sonic, and percussion holes. This chapter presents a discussion on pre-Midas Gold drilling followed by a discussion of Midas Gold drilling.

10.3 PRE-MIDAS GOLD DRILLING

The extent of pre-Midas Gold drilling varies significantly across the district and is broken out by individual areas (Yellow Pine, Hangar Flats, West End, Historic Tailings, and Scout) for further discussion.

The availability of pre-Midas Gold drilling data has varied by operator, time period, and deposit. Midas Gold has reviewed and incorporated all pertinent and available data into its database. Incorporated data include: geologic logs, drilling recovery, assay values, surface and down-hole surveys, and relevant Quality Assurance/Quality Control (QA/QC) measures.

Geologic logging associated with pre-Midas Gold drilling varied in format between past operators. General logging procedures utilized paper logs showing both visual logs and written observations. Characteristics recorded included core, cuttings and sludge recovery, lithology, alteration, pertinent mineralogy, sulfide percentage, oxide percentage/intensity, structures, and often a space was reserved for assay values (Au, Ag, Sb, and W).

Drilling recovery varied by era of drilling. Early drilling by Bradley and USBM had poor recovery due to the drilling technology of the time. Core recovery from later operators, however, was much better with Pioneer, Hecla, and Superior showing moderate recovery (averages in the 60-70% range), El Paso and Ranchers showing better recovery (averages in the 70-80% range), and Barrick exceeding 90% recovery.

Data for QA/QC programs were available from some pre-Midas Gold operators and are discussed in further detail within Chapter 11.

10.3.1 Yellow Pine

Past drilling within the Yellow Pine mineralized area was conducted with multiple methods by a number of different companies (Table 10.2). The historic Bradley and USBM drilling used conventional core drills of the time to drill AX, EX and BX sized core. The Hecla, Superior, Ranchers and Barrick drilling used wire line core drills with core sizes, comparable to Midas Gold, including PQ, HQ, and NQ. The RC drilling was conducted with buggy, track, and truck-mounted drills under dry and wet drilling conditions. The RC drill typically used a down-hole hammer with a 5.5-inch bit. Samples were collected by both a center return bit and an above-hammer interchange, and then traveled up the center of the drill string so that minimal contamination could occur. Typically, the overburden in the mineralized area was very thin and only a short section of casing was required. According to existing drill logs, operators began plugging their drill holes in the mid-1980s, prior to that time there was no hole-abandonment remediation required for previous drilling.

Historic files do not always describe in detail the methods used for locating holes. However, the operation was an active mine during parts of the drilling and the drill logs, plan maps, and sections illustrate the surveying standards that existed at the time of exploration, development, and mining activity. Many survey records from pre-Midas Gold drilling do exist, are well preserved, and were utilized to construct the drill hole database. In addition, a significant number of survey control points, old adits and shafts, and pre-Midas Gold drill hole collars were located by Midas Gold and included in its surveys, providing increased confidence in the location of pre-Midas Gold data including drill holes.

Table 10.2: Pre-Midas Gold Drill Holes in the Yellow Pine Deposit

Year	Operator	Type of Drilling	Holes	Feet
1939	USBM	Core	6	1,331
1940	Bradley	Core	248	52,867
	USBM	Core	46	14,759
1946	Bradley	Core	17	3,411
1949	Bradley	Core	2	870
1950	Bradley	Churn	9	1,386
		Core	3	825
1951	Bradley	Churn	6	272
		Core	14	4,133
1952	Bradley	Core	1	371
1953	Bradley	Core	3	1,068
1954	Bradley	Churn	10	894
1973	Twin Rivers Resources	Core	1	229
	Ranchers	Core	6	820
1974	El Paso	Core	1	400
1978	El Paso	RC	1	50
	Superior	Air Track	4	158
		Rotary	16	2,027
1982	Ranchers	Core	63	12,196
1983	Ranchers	Rotary	26	5,580
1984	Ranchers	Core	9	1,193
		RC	55	7,850
1986	Pioneer	Air Track	4	275
		Percussion	5	845
		RC	2	450
1987	Hecla	RC	29	1,080
1988	Hecla	RC	63	13,584
	Pioneer	RC	1	380
1989	Hecla	Core	2	593
		RC	21	2,150
1991	SMI	RC	71	2,167
1992	Barrick	Core	14	11,427
		RC	3	1,655
1997	SMI	RC	6	640
		Totals	768	147,936

Notes:
(1) For clarity the numbers in the table have been rounded to the nearest whole number.

10.3.2 Hangar Flats

Past drilling within the Hangar Flats mineralized area was conducted with multiple methods by a number of different companies (Table 10.3). Drill core sizes, by pre-Midas Gold operators included AX, EX, BX and NX and were reduced as drilling conditions required. Typically, the overburden in the mineralized area was very thin and only a short section of casing was required. According to existing drill logs operators began plugging their drill holes in the mid 1980's, prior to that time there was no hole-abandonment remediation required for previous drilling.

Historic files do not always describe in detail the methods used for locating holes. However, the operation was an active mine, and the drill logs, plan maps, and sections illustrate the surveying standards that existed at the time of exploration, development, and mining activity. Many survey records from previous drilling and underground development work by Bradley, as well as later campaigns under contract to the Defense Minerals Exploration Administration (DMEA) do exist, are well preserved, and were utilized to digitize the historic underground development workings and catalog drill data. Several of the older 1940's drill hole collars are still preserved and were surveyed and found to be within 3 - 6 ft of their expected locations. However, most collars were typically not preserved. Most of the later generation of drill holes, completed by Hecla in the area during the late 1980's, were located and surveyed in 2009 and 2010 and were found to be accurate to within 10 - 20 in.

Table 10.3: Pre-Midas Gold Drill Holes in the Hangar Flats Deposit

Year	Operator	Type of Drilling	Holes	Feet
1929	Bradley	Core	10	5,586
1940	Bradley	Core	28	6,207
1946	Bradley	Core	1	250
1947	Bradley	Core	3	961
1948	Bradley	Core	7	2,765
1952	USBM	Core	4	1,141
1953	Bradley	Core	4	2,448
	USBM	Core	8	2,528
1954	Bradley	Core	1	703
	USBM	Core	11	1,752
1955	USBM	Core	4	357
1974	El Paso	Core	1	399
		Rotary	1	200
1975	El Paso	Core	3	833
1982	Superior	Air Track	8	412
1988	Hecla	RC	5	935
1989	Hecla	RC	17	2,900
		Totals	116	30,377

Notes:
(1) For clarity the numbers in the table have been rounded to the nearest whole number.

10.3.3 West End

Past drilling within the West End mineralized area was conducted with multiple methods by a number of different companies, all of which were reputable industry operators or contractors (Table 10.4). Core drilling was much less common than RC and Air Track drilling, but core sizes included HQ and NX. The RC drilling was conducted with buggy, track, and truck-mounted drills under dry and wet drilling conditions. The RC drill typically used a down-hole hammer with a 5.5-inch bit. Sample was collected through both center return hammers and also with conventional above-hammer RC interchanges, and then traveled up the center of the drill string so that minimal down hole and cross contamination could occur. Typically, the overburden in the mineralized area was very thin, and only a short section of casing was required. According to existing drill logs operators began plugging their drill holes in the mid 1980's, prior to that time there was no hole-abandonment remediation required for previous drilling.

Historically, a drill location was first laid out by the mine surveyors with a specified easting and northing, and then a drill pad was constructed. After the pad was completed, the collar point was re-established. Original surveyor's records for most of the pre-Midas Gold drill holes are well preserved, and surveyed coordinates were verified against logs, as well as the dataset used in the resource models. Pre-Midas Gold drill hole collars were typically not preserved due to post-drilling mining operations in the area but some collars have been located by Midas Gold in its surveys and found to be accurate to within 3-15 ft. with some exceptions.

Table 10.4: Pre-Midas Gold Drill Holes in the West End Deposit

Year	Operator	Type of Drilling	Holes	Feet
1940	Bradley	Core	2	370
1973	Twin Rivers Resources	Core	4	1,167
1975	Superior	Core	2	607
1976	Superior	Core	17	6,661
		RC	12	1,080
1977	Superior	Air Track	58	4,995
		Core	24	6,618
1978	El Paso	RC	1	100
	Superior	Air Track	92	8,990
		RC	50	9,608
1981	El Paso	Air Track	35	1,660
	Superior	RC	9	1,750
1982	Superior	Air Track	26	1,131
1983	Superior	RC	45	11,219
		Rotary	28	3,124
1984	Superior	RC	12	2,653
1986	Pioneer	RC	31	6,913
1987	Pioneer	Air Track	8	470
		RC	76	16,670
1988	Pioneer	RC	48	20,180
1989	Pioneer	RC	79	32,939
1990	Pioneer	RC	32	9,200
1991	Pioneer	RC	32	11,615
1992	Pioneer	RC	42	11,240
1995	SMI	Core	2	305
1996	SMI	Core	1	374
		RC	59	20,780
1997	SMI	RC	62	15,840
		Totals	889	208,260

Notes:
(1) For clarity the numbers in the table have been rounded to the nearest whole number.

10.3.4 Historic Tailings

Pre-Midas Gold drilling within the Historic Tailings area was conducted primarily for water quality monitoring purposes. Stibnite Mines Inc. is the only known operator to have drilled in this area and they used both RC and auger drilling techniques. They drilled 25 holes totaling 1,512 ft in 1994 and 1996.

10.3.5 Scout

Past drilling in the Scout area was conducted with multiple methods by a number of different companies (Table 10.5). Bradley generally drilled AX, EX and BX core while Pioneer and El Paso drilled BQ, BX, NX, and HQ. According to existing drill logs the overburden thickness in this area is variable and in some instances operators were forced to abandon drill holes as a result. There was no hole-abandonment remediation required at the time of the previous drilling.

Historic files do not always describe in detail the methods used for locating holes, but conventional survey methods tied to existing ground control were typically utilized. However the drill logs, plan maps, and sections illustrate the standards that existed at the time of exploration. Some of the pre-Midas Gold hole collars are still preserved and were surveyed and found to be within 3 - 6 ft of their expected locations. However, most collars were typically not preserved.

Table 10.5: Pre-Midas Gold Drill Holes in the Scout Area

Year	Operator	Type of Drilling	Holes	Feet
1947	Bradley	Core	3	660
1948	Bradley	Core	1	405
1954	Bradley	Core	4	1,532
1955	Bradley	Core	2	1,123
1974	El Paso	Core	3	1,148
1975	El Paso	Core	2	1,289
1978	Superior	Air Track	1	40
1990	Pioneer	RC	2	715
		Totals	18	6,912

Notes:
(1) For clarity the numbers in the table have been rounded to the nearest whole number.

10.3.6 Pre-Midas Gold Coordinates and Grid Conversions

Three common local mine grids were used for surveying hole locations by pre-Midas Gold operators: the Bradley, Ranchers, and Hecla grids. Some other grids were used, but they were able to be converted into one of the main three grid systems. Each of the three grid systems had a known conversion into 1927 Idaho State Plane.

Midas Gold has used two separate methods for grid conversion from historic coordinate systems. From the Project inception until 2013, coordinates were converted by first converting historic coordinates into the Hecla grid, then into 1927 Idaho State Plane, and finally into Universal Transverse Mercator North American Datum of 1983 (UTM NAD83) using software (Trimble Surveying Pathfinder and Aspen Software, Golden Software Surfer and/or ESRI ArcGIS). In 2013, Midas Gold contracted Russell Surveying, Inc., a licensed and registered professional surveyor in Idaho to create conversions from various grid systems directly into NAD83 UTM coordinates. Converted UTM collar coordinates were then plotted and compared to historic maps for hole placement and consistency. Several errors were addressed and in every instance the source of the error was discovered and corrected. The Midas Gold database uses the converted coordinates for plotting purposes, but retains original information for any future verification purposes.

10.4 MIDAS GOLD DRILLING

In addition to the drilling performed by previous operators, Midas Gold has drilled 550 holes totaling approximately 326,274 ft of drilling distributed throughout the district during 2009 through 2013 (Table 10.6). Out of these holes, 506 were drilled within the current mineralized areas totaling approximately 316,039 ft. Drilling within the deposits by Midas Gold was planned to satisfy three main goals: (1) exploration of the Hangar Flats and West End mineralized areas (2) to confirm pre-Midas Gold drilling data and (3) to add infill drilling in the main deposit areas. Holes were drilled with RC, core, sonic, air track, and auger both as vertical and angle holes. Drill core sizes were generally HQ and NQ with a few PQ holes. Drill hole azimuths varied in all directions with dips ranging from -20° to -90°. The RC drill typically used a down-hole hammer with a 5.5-inch bit. Samples were collected through both center return hammers and also with conventional above-hammer RC interchanges, and then traveled up the center of the drill string so that minimal down hole and cross contamination could occur.

At Yellow Pine, the drilling has defined a large zone of anomalous gold, antimony and silver mineralization within the MCFZ and adjacent intrusive and, local sedimentary units. The mineralization is interpreted to follow two main orientations controlled by the MCFZ with related fractures and faults, and east-striking, high angle intersecting faults. The drill holes are generally located in a wide range of orientations with approximately 80 – 160 ft spacing within the deposit. They are typically oriented to the southeast or northwest, inclined steep to moderate. This orientation provides an oblique angle of intersection between the predominant planes of mineralization and the drill hole. Based on the wide range of drill hole orientations, some of the sample lengths do not represent true thickness of mineralization. In general, the drill hole intercept length is equal to or less than the true thickness of mineralization, as mineralization is broadly disseminated over significant widths and many drill holes bottom in mineralization (refer to Figure 7.5).

At Hangar Flats, drilling has defined a large zone of anomalous gold, antimony and silver mineralization within the MCFZ and adjacent intrusive units. The mineralization is interpreted to follow two main orientations controlled by both the MCFZ and northeast striking low angle splay faults. The drill holes are generally located in a wide range of orientations with approximately 100 - 210 ft spacing. The holes typically bear to the east or west, with some to the south, and are generally steeply to moderately inclined. This orientation provides an oblique angle of intersection between the predominant planes of mineralization and the drill hole. Based on the wide range of drill hole orientations, most of the sample lengths do not represent true thickness of mineralization. In general, the drill hole intercept length is greater than the true thickness of mineralization (refer to Figure 7.6).

At West End, drilling has defined a large zone of anomalous gold mineralization within the WEFZ and adjacent lithologic units. The mineralization is interpreted to follow two main orientations controlled by both the fault planes and stratigraphy. The drill holes are generally arranged in parallel at 65 - 100 ft spacing on section lines and inclined steeply to the northwest along parallel sections 100 ft apart. This orientation provides a high angle of intersection between the predominant structural plane of mineralization and the drill hole. Based on the wide range of drill hole orientations, most of the sample lengths do not represent true thickness of mineralization. In general, the drill hole intercept length is greater than the true thickness of mineralization (refer to Figure 7.7).

In the Historic Tailings, drilling has defined a flat-lying zone of fine-grained mine tailings of potentially economic grade. Drilling was completed with an auger rig using vertical holes with approximately 230 ft spacing which cross-cut the tailings perpendicular to the body. Due to the horizontal body of tailings being drilled by vertical holes, drill hole intercepts represent true thickness.

At Scout, drilling has defined a north/south trending near vertical zone of antimony, gold and silver mineralization within and adjacent to the near vertical Scout fault. Mineralization is hosted within fracture zones and is not constrained to a specific lithology. The majority of drilling is widely spaced (approximately 275 – 400 ft) and is oriented to the east in attempts to drill across the main mineralized zone to obtain true thickness. In general, the drill hole intercept length is greater than the true thickness of mineralization.

Table 10.6: Drilling by Area Completed by Midas Gold

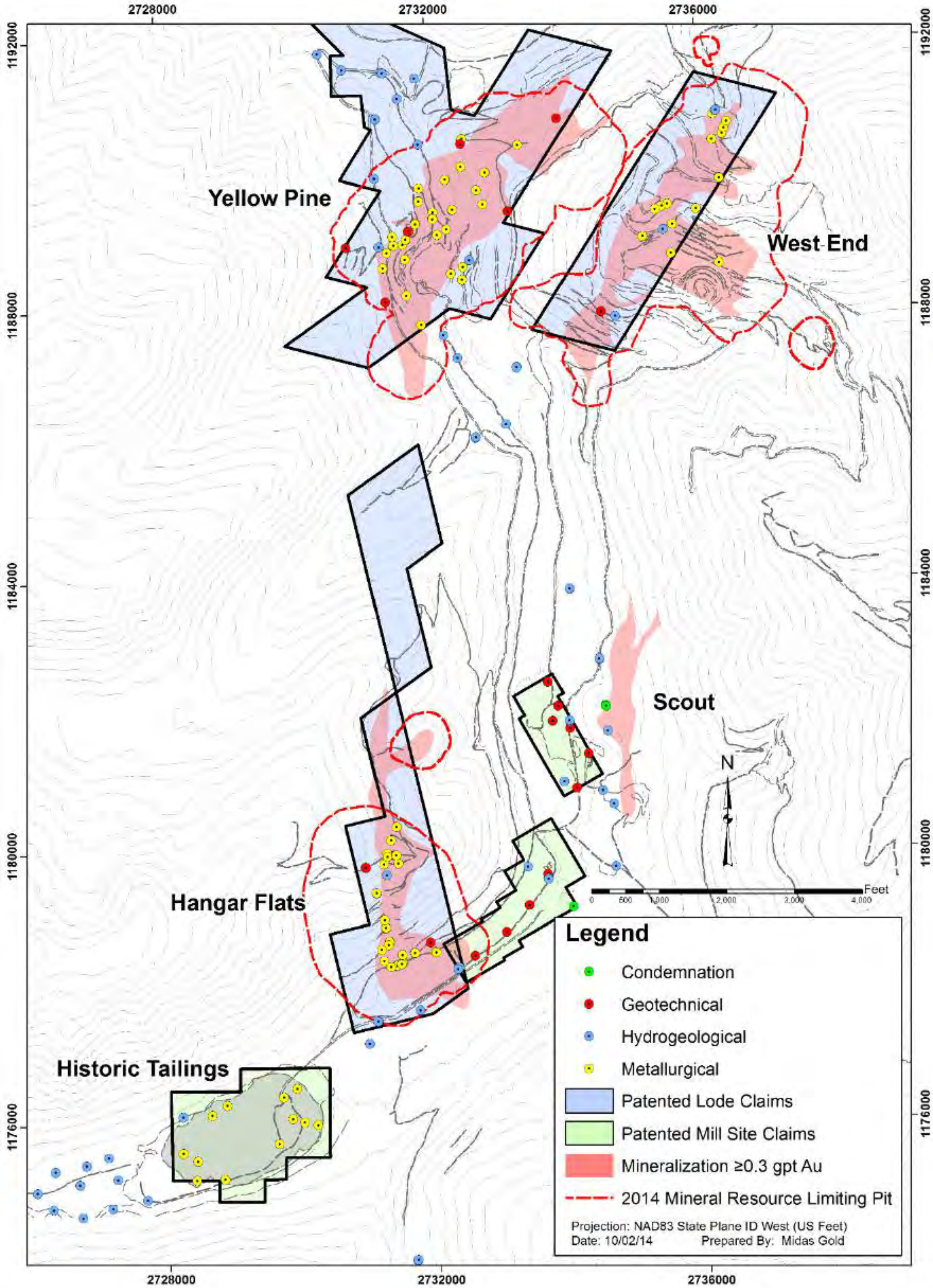
Hole Type	Year	# Holes	Feet
Yellow Pine			
Core	2011 - 2013	168	121,233
RC	2011 - 2012	62	29,312
Sonic	2011 - 2012	9	1,050
Air Lift	2012	3	414
	Totals	242	152,009
West End			
Core	2010, 2012 - 2013	34	28,837
RC	2011 - 2012	18	9,493
Air Lift	2012 - 2013	3	577
	Totals	55	38,907
Hangar Flats			
Core	2009 - 2013	102	86,798
RC	2012	23	15,675
Sonic	2011 - 2012	10	2,286
Air Lift	2012	6	949
	Totals	141	105,708
Historic Tailings			
Sonic	2011 - 2012	4	520
Air Lift	2012	1	60
Auger	2013	42	3,206
	Totals	47	3,786
Scout			
RC	2011 - 2012	5	4,310
Core	2012 - 2013	16	11,319
	Totals	21	15,629

Notes:
(1) For clarity the numbers in the table have been rounded to the nearest whole number.

10.5 GEOTECHNICAL AND HYDROLOGICAL DRILLING

Several of the previous operators conducted geotechnical and hydrological drilling for various purposes and many of their records still exist. The existing geotechnical data has been used by Midas Gold for initial planning purposes and several of the previous wells are still being utilized for water supply and monitoring purposes. More recently Midas Gold drilled a total of 13 bedrock geotechnical holes, 10 soil geotechnical holes, and 62 holes for water monitor well installations (Figure 10.7). Four other core holes were also used for multi-level sampler installations for hydrogeological purposes. Technical consulting firms were contracted to plan and manage the geotechnical and hydrogeological programs. SRK Consulting was hired to oversee the geotechnical and hydrogeological drill campaign in 2011 - 2012 and MWH Global took over the hydrogeological campaign in 2012 - 2013.

Figure 10.7: Hydrological, Geotechnical, Metallurgical, and Condemnation Drilling



Several of the holes drilled for geotechnical analysis in soils were also used to install water monitor wells; similarly, several of the holes drilled for water monitoring wells were also used for geotechnical analysis. For example, the hydrogeological holes shown on Figure 10.7 around the Historic Tailings area are currently being used for water monitoring purposes and are shown as such; however, these holes were drilled specifically to evaluate the geotechnical conditions beneath the potential Tailings Storage Facility. Geotechnical holes included bedrock core holes and overburden soil holes drilled with core, sonic, air track, and auger rigs. Water monitor wells were drilled with core, sonic, and RC rigs.

Core for the 13 bedrock geotechnical holes was drilled with split tubes and oriented using a Reflex ACT II tool. Whenever the drill was operating, geologists were onsite to log geotechnical data as the core was retrieved from the hole. Piezometers were installed down six of these holes and are presently collecting down hole data (Figure 10.7).

Monitor well holes included bedrock and alluvial wells. Well installation was carried out by SRK Consulting and MWH Global. Many of the water monitor wells were also logged for soil geotechnical purposes.

Ten auger holes were drilled in the vicinity of the potential plant site. These holes were drilled and logged solely for geotechnical purposes to aid in infrastructure construction considerations.

10.6 METALLURGICAL DRILLING

Midas Gold drilled 11 holes specifically to collect samples for metallurgical sampling. Core size for these holes was PQ but was sometimes reduced to HQ as drilling conditions required. Quartered core from these holes was assayed for use in mineral resource estimation and the remainder submitted or retained for metallurgical work with a portion archived in the Midas Gold core storage facilities.

Additionally, samples were taken from 99 other core and auger holes to be used for metallurgical testing (Figure 10.7). These holes were generally drilled with HQ core and were selected to provide representative samples from each of the advanced stage deposits (Yellow Pine, West End, Hangar Flats, and Historic Tailings).

10.7 CONDEMNATION DRILLING

Condemnation drilling completed by Midas Gold focused on two specific areas of potential infrastructure and consisted of 4 holes (Figure 10.7). These holes were drilled with HQ tooling. Three holes were drilled at Scout from the same drill pad near the current shop and camp as dual-purpose condemnation/exploration testing a potential deposit. A 9-hole exploration drill program followed in this area. A single hole was also drilled south of Scout within potential infrastructure areas for condemnation.

10.8 GEOLOGICAL LOGGING

Geologic logging performed by Midas Gold utilized paper log sheets in 2009 - 2010 and digital logging methods in 2011 - 2013. In 2009 and 2010, geologic logging on paper was completed onsite after core was received from the drillers. Logs included both visual and written observations recording lithology, alteration, pertinent mineralogy, sulfide percentage, oxide intensity, and structures. These paper logs were digitally captured after the 2009 and 2010 field seasons.

In 2011 - 2013, preliminary core logging was completed on site and detailed logging was completed at the core logging facility in Lake Fork, ID. Preliminary geological logging performed at Stibnite after core was received from drillers identified general geology and alteration for hole-tracking and daily reporting purposes. This was followed up with detailed geologic logging using Microsoft Access digital logging forms. Pertinent geologic observations were digitally recorded including recovery, rock quality, lithology, alteration, mineralization, and structures. The Microsoft Access form was also used to record sample intervals and basic header information including azimuth, inclination, survey coordinates, logging geologist, drilling contractor, etc. Once logging was completed for a hole, the completed log was added to Midas Gold's Microsoft Access database after data verification.

Reverse circulation chip logging in 2011 and 2012 was completed using paper logs either at the drill rig or at the Stibnite core facility. These paper logs were later entered digitally using Microsoft Access logging forms and the logs were added to the database.

10.9 DRILLING RECOVERY

In general, both RC and core recovery was good for all drilling completed by Midas Gold. Core recovery averaged 92.6 %, and RC recovery was good to excellent. Whenever the RC drilling encountered voids, recovery suffered significantly, and if it could not be regained, the hole was terminated.

Numerous studies and statistical evaluations have been performed by Midas Gold staff testing the relationship between recovery and grade across the Project for both Pre-Midas Gold drilling and recent drilling conducted by Midas Gold. No significant relationship could be found.

Cyclicity issues were identified within a small number of the RC holes drilled by Midas Gold. Individual intervals were analyzed and those showing cyclicity were flagged for omission in mineral resource modeling. Problematic intervals were only identified and flagged in a small number of RC holes which were all drilled in 2011 and, as a result, these holes were excluded from resource estimation.

10.10 ROCK QUALITY DESIGNATION

Rock Quality Designation (RQD) is a measure of naturally occurring fractures in a rock and was applied to all core drilled by Midas Gold since the beginning of the 2010 drilling program, starting with hole MGI-10-12. Approximately 95% of Midas Gold's core drill holes were measured for RQD between 2009 and 2013. RQD was measured as the sum of all complete core fragments with lengths greater than 3.9 in (10 cm) in a given core run with > R1 hardness value (will not crumble under a firm blow with the point of a geologic hammer) over the length of the core run. Lengths were measured along the centerline of the core, ignoring fault gouge or other low competency material and paying close attention to mechanical breaks from drillers boxing the core, as these are not naturally occurring fractures.

10.11 DRILL HOLE COLLAR SURVEYS

During the 2009-2013 Midas Gold drilling programs, drill sites were located using handheld Global Positioning System (GPS) receivers. Drill hole orientations were calculated based on actual drill collar locations to ensure that holes were properly oriented. Alignment stakes were set and drill alignments surveyed using conventional survey tools or in some cases a Brunton-style compass.

Once holes were completed, the collar was marked with a cement cap containing a steel pin attached to a steel chain extending above ground surface with a tag identifying the drill hole number. Between 2009 - 2012, a professional surveyor was used to survey the final collar locations. In 2013, a high-precision GPS was utilized by onsite geologists to survey final drill hole collar coordinates with an estimated accuracy of ± 2 ft.

10.12 DOWN HOLE SURVEYS

Down hole surveys were performed on core holes using a number of survey instruments including: an acid etch clinometer, tropari, and a Reflex EZ-Shot tool to measure deviation from the collared orientations. Surveys were generally taken every 200 ft down hole with some exceptions due to lost or collapsed holes.

Survey values were received from drill contractors on paper logs and were entered into GEMS Logger by the logging geologist for each individual hole. Magnetic declination corrections were applied by GEMS logger based on the declination correction entered by the logging geologist. Declination corrections were modified at least annually based on changing magnetic declination.

10.13 SAMPLE LENGTH AND TRUE THICKNESS

Sample length was a set value for the RC (5 ft) and auger drilling (5 - 10 ft within spent ore material, 2 ft within tailings). For core drilling, sample length was determined by the geological relationships observed in the core and was generally 5 - 7.5 ft. Changes in lithology and mineralization were used as sample breaks, and regular sample intervals were used within lithologic units and intervals of similar mineralization intensity.

In general, the deposits show broadly disseminated mineralization with a few major structural boundaries. As these are not vein deposits, the term "true thickness" is difficult to apply due to the entirety of the rock mass being potential ore-grade material. However, an attempt was made to drill across major structures to test their effect on mineralization. Based on the wide range of drill hole orientations, many of the sample lengths do not represent true thickness of mineralization. In general, at Hangar Flats and West End the drill hole intercept length is greater than the true thickness of mineralization. At Yellow Pine, the drill hole intercept length is generally less than true thickness.

10.14 CORE, CUTTINGS, REJECT AND PULP STORAGE

Core and cuttings were received by Midas Gold personnel from the drilling contractors and remained under supervision until shipped to Midas Gold's core logging facility in Lake Fork, ID. Once in Lake Fork, core and cuttings were stored within the Midas Gold core logging facility which is supervised during the work day and locked when vacant (nights and weekends). After core was logged and sampled, the remaining halved core was stored either in Midas Gold's Lake Fork buildings, in Midas Gold's Cascade warehouse, or behind a secured chain-link fenced compound at the Cascade warehouse. Rejects were stored in the same locations. Once pulps were received back from the assay labs, they were stored by Midas Gold at the Lake Fork facility. Rejects are stored inside of the chain-link fence at the warehouse in Cascade. Additionally, some core being held for future metallurgical sampling is being stored in a secure refrigerated truck behind the chain-link fence in Cascade. All storage locations remain locked when no Midas Gold personnel are present and have restricted access. In Cascade, both the fence and the warehouse remain locked.

Throughout 2014 Midas Gold relocated their Lake Fork facilities. Employee office facilities were moved to their current location in Donnelly, ID. Consequently all core, pulp, rejects, and additional samples at their Lake Fork facility have been moved to secure indoor and outdoor storage at the Cascade warehouse detailed above. All storage locations remain locked when no Midas Gold personnel are present.

Going forward, Midas Gold intends for all samples to be cut and logged at their onsite facilities and stored at their indoor and outdoor Cascade warehouse facilities.

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11 SAMPLE PREPARATION ANALYSES AND SECURITY

This section provides an overview of the sample preparation, analyses, and security procedures used by Midas Gold; where available similar information is also provided for pre-Midas Gold activities.

11.1 SAMPLING METHODS

Throughout the last 85 years, multiple drilling and sampling methods have been used across the district by pre-Midas Gold operators as well as Midas Gold. Sampling methods have varied based on the era and the type of drilling.

11.1.1 Pre-Midas Gold Sampling

Early operators generally sampled drill core and sludge while later operators drilled and sampled either core or reverse circulation chips. Later operators used modern wire-line core drilling methods resulting in better core recovery. Reverse Circulation (RC) drill holes were drilled under both wet and dry conditions and samples collected from a cyclone or similar splitter. Sample lengths, regardless of the drilling campaign, were generally 5 ft in length, although many sample intervals were selected based on changes in lithology or changes in intensity of alteration and mineralization. Few documents have survived to describe sample preparation methods and little to no chain of custody records for previous operators are available.

11.1.2 Reverse Circulation Drill Sampling

Midas Gold RC holes were cased into competent bedrock and drilled wet. Samples were collected every five feet and holes flushed and cleaned between samples with water and drilling products. Sampled material was collected from a cyclone splitter into plastic totes. A flocculent was added if necessary and, after settling, the excess clear water was decanted off and the remaining sample was poured into labeled sample bags. QA/QC samples were inserted at the rig by the rig geologist and typically included 1 certified standard, 1 blank and 1 cyclone splitter reject every 100 ft. (i.e. every 20th sample). Sample bags were placed into larger rice bags which were placed into bulk storage sacks and shipped to Lake Fork, ID for shipping to the laboratory. Pre-numbered bar codes were utilized for sample numbering.

11.1.3 Core Drill Sampling

From the beginning of the core drilling program in 2009 through 2011, core was generally sampled on 5 ft intervals with sample breaks made at significant changes in lithology or intensity of alteration and/or mineralization. Beginning in 2012, sample intervals for core were based on the logging geologist's interpretation of the intensity of mineralization for example; if core was mineralized, samples were selected in 6.5 ft lengths; if core was not mineralized samples were selected in 7.5 ft lengths. The core logging geologist marked the core with a lumber crayon to provide a line for the core sawyer to split veins and joints into representative halves. Half of the cut core was placed into canvas sample bags, which were placed into labeled rice bags, and then placed into bulk storage sacks for shipment to the laboratory. Typically, sampling was conducted in batches of 40 samples including 2 certified standards, 2 blanks, and 2 quarter-core duplicates. Pre-numbered bar codes were utilized for sample numbering.

11.1.4 Sonic and Auger Drill Sampling

Sonic drilling samples were collected by the drilling contractor and placed into plastic sleeves which were set into cardboard boxes. This material was sampled in a manner similar to drill core samples.

Auger samples were collected in a split tube and split in half by the geologist. Material was composited into 10 ft samples within the SODA material and 2 ft samples within the tailings material and then placed into canvas sample bags. The other half of the tailings samples were retained and placed in wooden core boxes. In the Historic Tailings, at least one sample in 35 of the 42 drill holes was taken as a Shelby sample for specific gravity and particle size analysis. The geologist inserted one standard and one blank into the sample set for each hole within the tailings. The split tube was washed thoroughly between samples to prevent cross-contamination.

11.2 SECURITY AND CHAIN OF CUSTODY

All samples were kept under direct supervision of Midas Gold staff and its contractors or within locked facilities. Changes in custody were accompanied by signed and dated Chain of Custody (COC) forms.

RC and auger samples were bagged at the drill rig and prepped for shipment to the assay lab under supervision of the rig geologist. RC and auger samples were shipped to the Lake Fork logging facility in bulk storage bags accompanied by a signed COC form detailing drill hole numbers, footages, sample numbers, and the shipment date.

Drill core was picked up at the drill rig by the site geologist while performing the daily rig inspections. After inspecting the core boxes for errors, a COC was completed documenting the transfer of core from the rig to the Stibnite core shack. Often the initial COC would be documented on the driller's daily log and included the box numbers, footages, date, and geologist's name and signature. At the core shack, a summary log was completed to verify and record box numbers, footages, lithology, mineralization and other rock characteristics. Upon completion of the summary log, the core was prepared for shipping to the Lake Fork logging facility by Midas Gold staff or contractors. When shipped, core was accompanied by a signed COC form detailing the hole numbers, footages, box numbers, and shipment date.

Once the core or samples were received at the Lake Fork facility, the receiver checked the COC for errors and stored the core for future logging/sampling in a secured site which was locked when no personnel were present. Once detailed logging and sampling of core was complete, the samples were prepped for shipping, bagged in rice bags, and taped shut with tamper-proof security tape. Each shipment was accompanied by another COC form to the assay lab. Upon receipt, the lab then verified that the security tape was undisturbed and completed the COC form.

11.3 DENSITY

In 2010, Midas Gold sent 61 samples from the 2009 and 2010 drilling campaign to ALS Chemex Labs, Ltd. (ALS) for density determination using a paraffin wax coating. Beginning in 2011, density measurements for core material were determined using hydrostatic weighing. Measurements were collected by Midas Gold geologists on approximately 0.5 ft core intervals every 50-200 ft downhole, or within different lithologic units, totaling 3,196 intervals. Four hundred ninety-six of these density samples were also submitted to ALS for density determination with paraffin wax coating. ALS results compared to measurements by Midas Gold showed a root mean squared coefficient of variation (RMS CV; a statistical tool routinely used to determine precision through using the quadratic mean of the relative standard deviation for each pair) of 0.98%, indicating there was no significant difference between the in-house measurements and third-party, independent certified lab results for density.

For the unconsolidated material within the Historic Tailings, 35 samples were sent to Strata Geotechnical Testing Laboratories in Boise, ID for density determination using the ASTM D2937 method. This method involves collecting an in-situ sample using a drive-cylinder with a known volume, weighing the sample, and calculating the density of the collected material.

11.4 ANALYTICAL LABS AND METHODS

There is little documentation of the sample preparation, analysis, and security for most samples from pre-Midas Gold operators. USBM utilized a government laboratory and analyzed drill core and sludge using a conventional 30 g fire assay pre-concentration method followed by gravimetric analysis. Other operators used several assay laboratories (both for primary and check assays) with CN-leach assays followed by atomic Absorption (AA) for oxide mineralization and conventional fire assay techniques for sulfide mineralization. Bradley drilling sludge samples were analyzed using conventional fire assay techniques in company owned Yellow Pine and Boise laboratories. Table 11.1 shows the various analytical labs used by different operators. The various analytical methods utilized at various laboratories by pre-Midas Gold operators had different lower detection limits, upper reporting limits and sensitivities which are documented in the company's database and archives.

Table 11.1: Off-Site Assay Laboratories Used by Pre-Midas Gold Operators

Laboratory	Location	Operator	Year
T.S.L. Laboratories Limited	Spokane, WA, USA	El Paso	1973, 1978
		Superior	1975-1978, 1981
Union Assay	Salt Lake City, UT, USA	Ranchers	1973, 1975-1978 1982, 1984
Bondar Clegg	BC, Canada	Superior	1976
	North Vancouver, BC, Canada	SMI	1995-1996
Rocky Mountain Geochemical Corp.	Midvale, UT, USA	Superior	1976-1977
	Reno, NV, USA	Ranchers	1983-1984
Monitor Geochemical Laboratory	Elko, NV, USA	Superior	1978
Hazen Research	Golden, CO, USA	Ranchers	1982
Peter Mack	Wallace, ID, USA	Ranchers	1982
South Western Assayers and Chemists	Tucson, AZ, USA	Ranchers	1982
Mountain States Research and Development	AZ, USA	Ranchers	1982-1984
Silver Valley	Osburn, ID, USA	Superior	1983
Hunter	Sparks, NV, USA	Pioneer	1986-1988
ALS Chemex Labs Inc.	N. Vancouver, BC, Canada	Hecla	1989
		Barrick	1992
SVL Analytical Inc.	Kellogg, ID, USA	SMI	1997

11.4.1 Assay Laboratories

Midas Gold utilized multiple laboratories for assay, check assay, and metallurgical work in both the US and Canada. All labs were ISO 17025 or 9001 certified. Table 11.2 summarizes the assay laboratories used by Midas Gold for sample analysis from 2009 to 2014. A total of four labs have been used in the United States and Canada for primary and check assays.

Table 11.2: Analytical Laboratories Used by Midas Gold

Laboratory	Location	Certification/ Accreditation	Use	Year
ALS	Elko, Reno, and Winnemucca, NV, USA; Vancouver, BC, Canada	ISO 17025:2005 ISO 9001:2008	Primary Lab 2009-Present	2009 - 2014
American Analytical Services (AAS)	Osburn, ID, USA	ISO 17025	Check Assays	2010, 2012-2013
Inspectorate	Reno, NV, USA	ISO 9001:2008	Check Assays	2009, 2012-2014
SGS Canada, Inc.	Vancouver, BC, Canada	CAN-P-1579 17025:2005	Check Assays	2014

11.4.2 Metallurgical and Geochemical Laboratories

Table 11.3 summarizes the laboratories used by Midas Gold for analysis from 2010 to 2014. A total of eight labs have been used in the United States and Canada for metallurgical and geochemical testing.

Table 11.3: Metallurgical and Geochemical Testing Laboratories Used by Midas Gold

Laboratory	Location	Certification/Accreditation	Use	Year
SGS Canada, Inc.	Lakefield, ON, Canada; Vancouver, BC, Canada	CAN-P-1579, CAN-P-1587, CAN-P-4E (ISO/IEC 17025:2005)	Primary Metallurgical Testing Lab	2010-2014
Kingston Process Metallurgy, Inc.	Kingston, ON, Canada	n/a	Metallurgical Testing	2013-2014
Pocock Industrial, Inc.	Salt Lake City, UT, USA	n/a	Metallurgical Testing	2013-2014
McClelland Laboratories	Sparks, NV, USA	EPA ID #: NV00933	Geochemical Testing	2012-2014
Western Environmental Testing Laboratory	Sparks, NV, USA	EPA ID #: NV000925	Geochemical Testing	2012-2014
SVL Analytical	Kellogg, ID, USA	EPA ID #: ID000019	Geochemical Testing	2013-2014
Inter-Mountain Laboratories	Sheridan, WY, USA	EPA ID #: WY00005	Geochemical Testing	2014
Dynatec Labs	Fort Saskatchewan, Alberta, Canada	ISO/IEC 17025; 2005	Metallurgical Testing	2012

11.5 SAMPLE PREPARATION AND ANALYSIS

Midas Gold samples were received and weighed by the primary assay lab. Core samples were prepared based on laboratory specifications which involved being pulverized to 70% passing a ¼ inch mesh (6 mm) and dried at a maximum of 140 degrees Fahrenheit (60 degrees Celsius). Dried material was split and crushed to 70% passing No. 10 mesh, split again, and pulverized to 85% passing No. 200 mesh. Material passing through the No. 200 mesh was then run with four primary analytical techniques.

Multi-element analysis was done by a 4-acid digestion followed by inductively coupled plasma atomic emission spectroscopy (ICP-AES) for 33 elements with an Hg add-on. Every 20th sample was digested in aqua regia followed by an inductively coupled plasma mass spectrometry (ICP-MS) finish for 51 elements with a fluorine add-on. Arsenic had a 5 parts per million (ppm) lower detection limit and a 10,000 ppm upper reporting limit. Samples reporting > 10,000 ppm As were re-analyzed by using a digestion in 75% aqua regia followed by an ICP-AES finish with a lower detection limit of 0.01% and an upper reporting limit of 60%. Antimony had a 5.0 ppm lower detection limit and a 10,000 ppm upper reporting limit. Samples reporting values > 500 ppm Sb were re-analyzed using 0.9 g sample added to 9.0 g Lithium Borate flux mixed well and fused in an auto fluxer. A disc was prepared from the melt and analyzed using X-ray fluorescence spectroscopy with a lower detection limit of 0.01% (100 ppm) and an upper reporting limit of 50%. Sulfur had a 0.01% lower detection limit and a 10% upper reporting limit. Samples reporting values > 2% S were re-analyzed by using a 0.01 – 0.1 g sample in a Leco sulfur analyzer using an Infrared (IR) detection system with a 0.01% lower detection limit and a 50% upper reporting limit.

All gold assays were performed using a 30 g fire assay charge followed by an atomic absorption spectroscopy (AAS) finish with a 0.005 ppm lower reporting limit and a 10 ppm upper reporting limit. Samples reporting values > 6 ppm were re-analyzed using a 30 g fire assay charge followed by a gravimetric finish with a 0.05 ppm lower reporting limit and a 1,000 ppm upper reporting limit. Samples reporting values >10 ppm were analyzed by metallic screen method with a 0.05 ppm lower reporting limit and a 1,000 ppm upper reporting limit.

Silver was analyzed via the initial multi-element ICP-AES analysis with a 0.5 ppm lower detection limit and a 100 ppm upper reporting limit. Samples reporting values > 10 ppm Ag were reanalyzed using a 4-acid digestion followed by an ICP-AES or AA finish with a 1.0 ppm lower detection limit and a 1,500 ppm upper reporting limit. Samples reporting values > 750 ppm Ag were reanalyzed using a 50 g fire assay charge followed by a gravimetric finish with a 5 ppm lower detection limit and a 10,000 ppm upper reporting limit.

11.6 DATABASE VERIFICATION

The Midas Gold database administrator has multiple measures in place to check the database for errors. Interval verification tools are run regularly to check for intervals that are overlapping or out of sequence. Digital assay data are received from the primary assay laboratory and are imported directly into the database. Assay data in the database are periodically verified against a master assay spreadsheet and original laboratory analytical reports to prevent assay value errors. Furthermore, sample number ranges are examined for unreasonable differences that may indicate sample switches or typing errors.

As part of the development of the PEA, SRK checked 88% (28,692 records) of the assay intervals in the database versus original data files delivered by ALS, from the first drill hole, MGI-09-01, to MGI-12-210 (210 total holes) and found 14 errors in assay data which were all corrected. Also as part of the PEA, SRK verified the core logging information in the database of 21 random holes versus the original logs as filled out by the core logger. Post PEA database verifications are summarized in Section 12.

11.7 QUALITY ASSURANCE AND QUALITY CONTROL

Pre-Midas Gold operators conducted various QA/QC programs for both their drilling and mine assay operations. Some records of QA/QC measures may not have survived to be reviewed by Midas Gold. However Section 11.7.1 details the records that Midas Gold has collected and catalogued.

Midas Gold exercised strict and rigorous QA/QC protocols throughout the different drilling campaigns. Periodically these protocols were assessed for adequacy and improved accordingly.

11.7.1 QA/QC Pre-Midas Gold

Pre-Midas Gold operators had varying QA/QC programs, but not all records have survived. QA/QC data available for each operator from existing records are detailed in Table 11.4, where insertion rate is detailed for drilling conducted by that operator. Historic reports indicate that Bradley used duplicates and standards as QA/QC measures at Hangar Flats, but exact insertion rates are unknown.

Table 11.4: Pre-Midas Gold QA/QC Measures and Insertion Rates

Company	Deposit	Check(2)	Reject(3)	Rerun(4)	Standard	Blank	Totals(1)
Pioneer	West End	1.74%	5.54%	0.07%	8.67%	-	16.02%
SMI	West End	2.00%	-	2.56%	1.27%	0.35%	6.18%
Superior	West End	10.57%	-	0.56%	1.25%	-	12.38%
Pioneer	Yellow Pine	-	-	-	18.35%	-	18.35%
Ranchers	Yellow Pine	4.42%	6.44%	-	-	-	10.86%
Superior	Yellow Pine	1.19%	-	-	-	-	1.19%
Barrick	Yellow Pine	3.88%	-	-	-	-	3.88%

Notes:
(1) Percentages stated are based on QA/QC analyses recovered from historical files and are likely not comprehensive.
(2) Check assays were performed at third party laboratories.
(3) Rejects consisted of a combination of sample rejects and sludge samples run at internal and third party laboratories.
(4) Rerun assays were performed at internal laboratories.

11.7.2 QA/QC by Midas Gold (2009-2012)

Midas Gold initially created a QA/QC program in 2009 to provide adequate confidence in the data collection and processing. As part of the development of the PEA, SRK consulting examined the performance of Midas Gold's QA/QC program from 2009 through June 2012. SRK determined that performance of certified blanks was good but the locally sourced in-house blanks performed poorly. No significant bias was determined to be present in core duplicates. RC field rejects were determined to exhibit no significant bias and good reproducibility. No evidence of systematic analytical bias was observed in standards.

11.7.3 QA/QC by Midas Gold (2012-2013)

Following review by SRK in 2012, Midas Gold revised its QA/QC program. New protocols incorporated additional blank materials and certified antimony standards, discontinued use of non-certified reference materials, increased insertion rates for antimony standards, increased use of second lab check assays, and initiated use of blind reject samples. Table 11.5 shows the insertion rates of various QA/QC measures used in Midas Gold drilling since the beginning of 2012, which may overlap slightly with the period reported in the PEA. QA/QC measures are described in sections below. Insertion rates were determined within mineralized zones that were defined by gold grades greater than 0.5 g/t.

Table 11.5: Midas Gold QA/QC Measures and Insertion Rates within Mineralized Zones

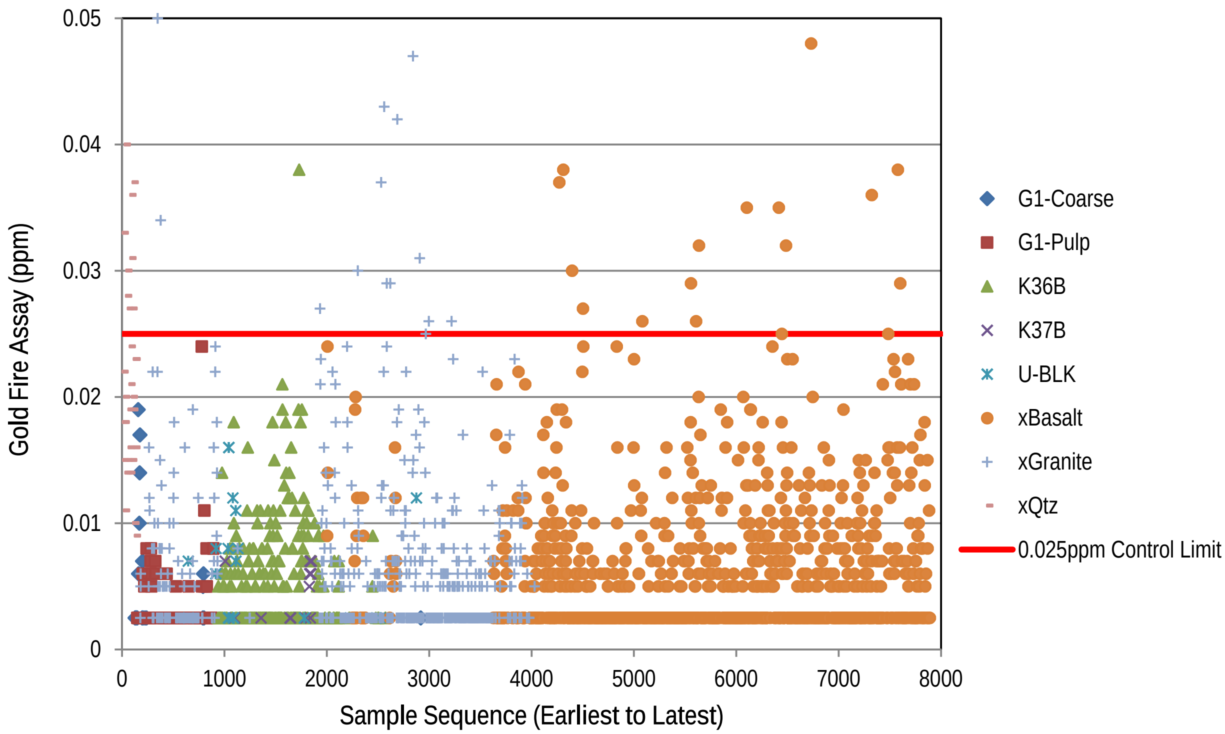
Year	Deposit	Assays	Field Duplicates	Pulp Duplicates	Check	Reject	Standard	Blank	Totals
2012	Yellow Pine	1300	2.77%	5.38%	21.00%	3.62%	6.31%	4.92%	44.00%
	Hangar Flats	364	3.30%	3.57%	7.42%	4.40%	7.14%	5.77%	31.60%
	West End	406	4.93%	5.42%	14.53%	5.17%	5.42%	4.68%	40.15%
	Scout	643	4.98%	5.44%	1.87%	0.00%	6.53%	4.67%	23.49%
2013	Yellow Pine	1117	4.66%	5.64%	3.67%	3.22%	9.13%	4.39%	30.71%
	Hangar Flats	194	5.15%	5.15%	3.61%	3.61%	9.79%	4.12%	31.43%
	West End	64	4.69%	7.81%	6.25%	6.25%	4.69%	7.81%	37.50%
	Historic Tailings	557	0.00%	4.85%	8.44%	0.00%	7.36%	1.97%	22.62%
	Scout	9	0.00%	0.00%	0.00%	44.44%	0.00%	0.00%	44.44%

11.7.4 Blanks QA/QC

Midas Gold used a total of 2,374 blanks in the sample stream, 312 of which were certified (Figure 11.1). Non-certified in house blanks were composed of locally sourced, unmineralized quartzite, basalt, and granite.

Upon evaluation, blanks reporting values below 0.025 ppm Au were considered satisfactory. Certified blanks reported 100% of values under this limit and non-certified blanks reported 86% of values under this limit. The only blank material utilized since the PEA was locally sourced Miocene basalt which exhibited a failure rate of 1.1%, with two samples assaying and re-assaying above 0.5 ppm gold.

Figure 11.1: Blank Performance – Gold



11.7.5 Standard Reference Materials QA/QC

In post-PEA drilling, Midas Gold began to decrease the use of non-certified gold standards, as well as increase use of certified antimony standards. Insertion rate of standards typically exceeded 5% for drilling within all deposits. Midas Gold used a total of 1,588 certified gold standards, 1,021 non-certified gold standards, and 499 certified antimony standards (Figure 11.2, Figure 11.3). Some antimony standards were not certified at the time of use, but subsequently received certification.

Upon evaluation, standards reporting within two standard deviations of the expected value were considered satisfactory. Standards were flagged for evaluation when reporting between two and three standard deviations from the expected value and flagged as failed when reporting over three standard deviations. Standards flagged for evaluation were re-run on a case-by-case basis while the procedures for standards flagged as failed are described in Section 11.7.9. Certified gold standards reported 95% of values within satisfactory limits, non-certified gold standards reported 96% of values within satisfactory limits, and certified antimony standards reported 99% of values within satisfactory limits.

Figure 11.2: Certified Gold Standards

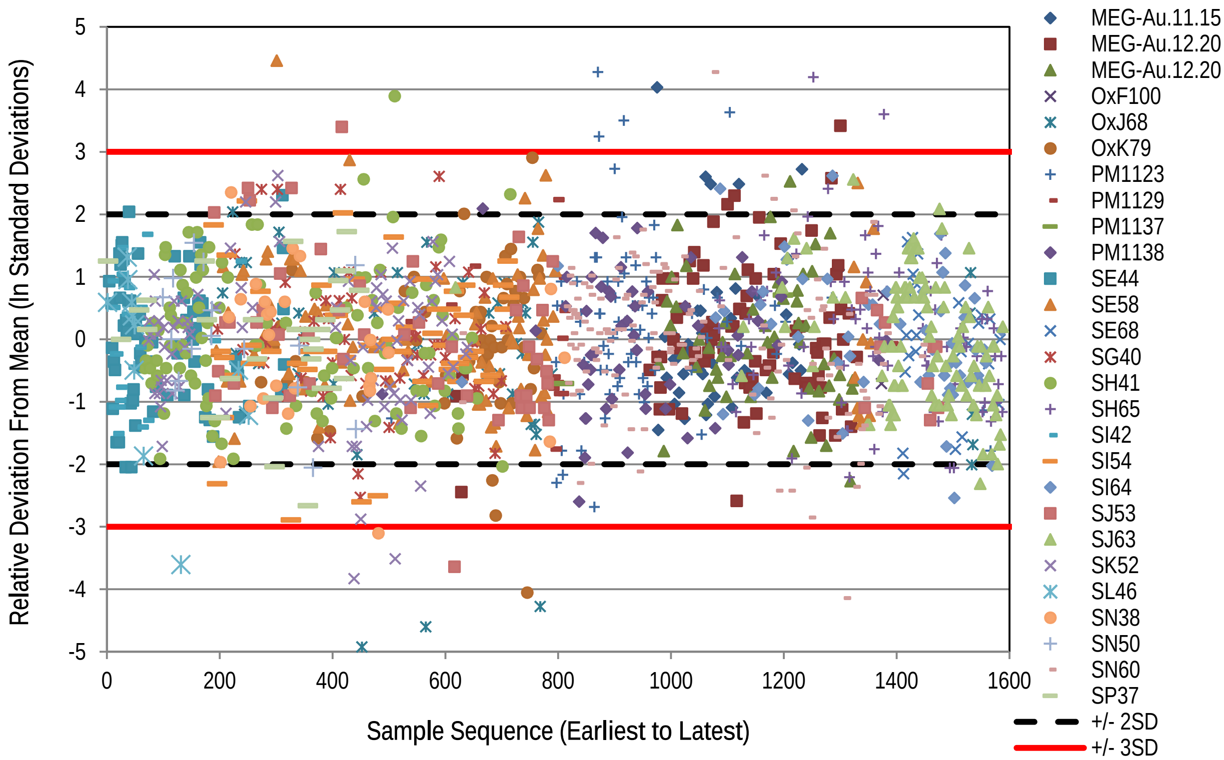
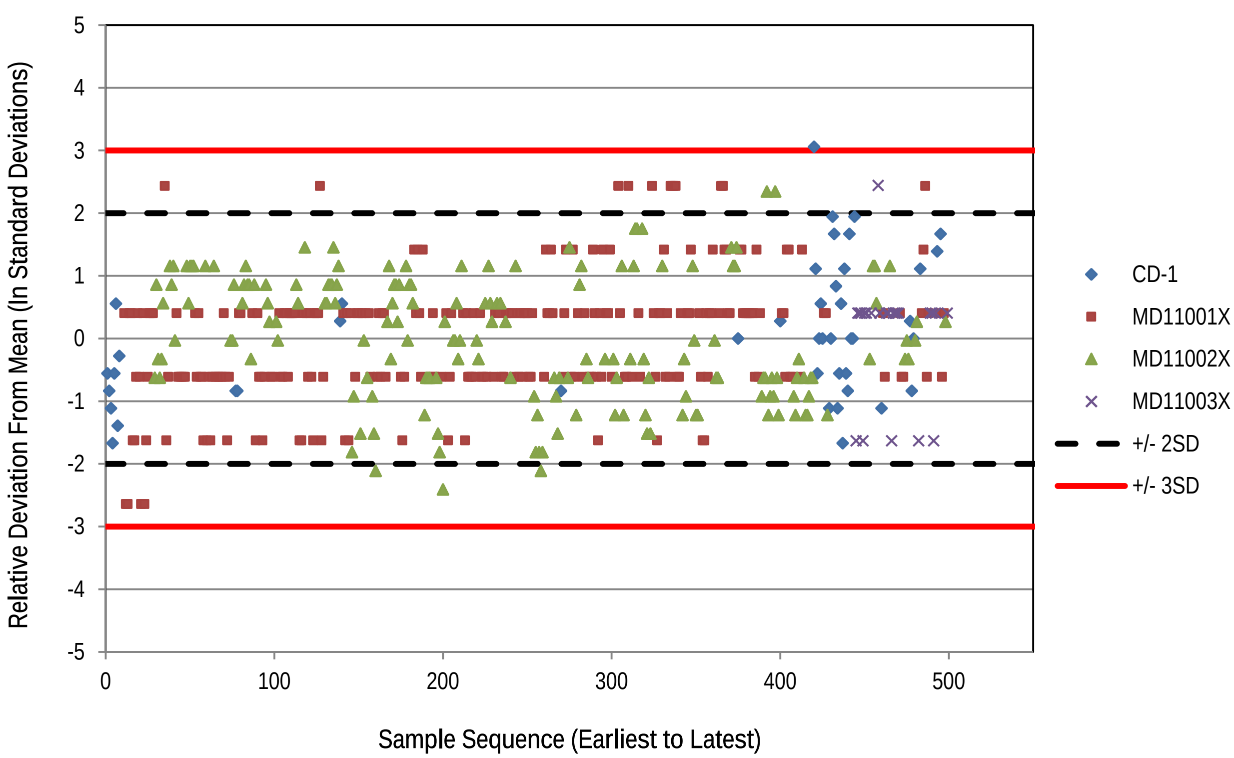


Figure 11.3: Certified Antimony Standards

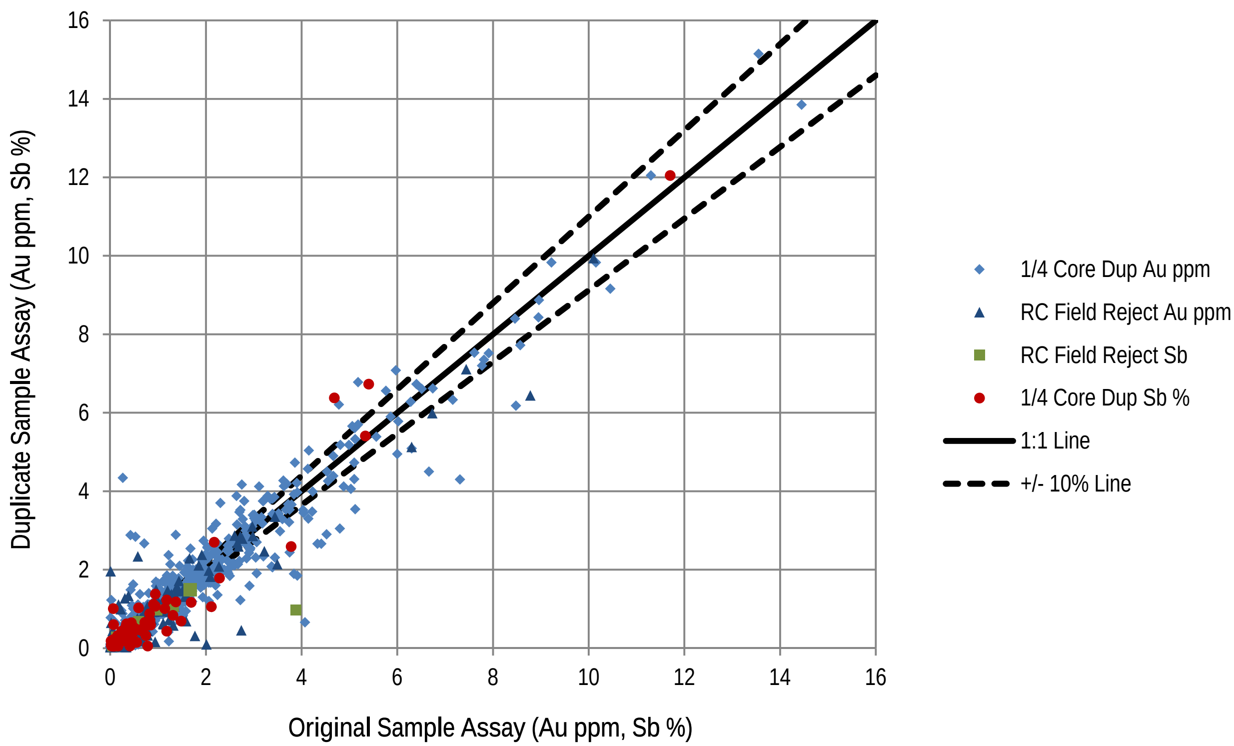


11.7.6 Field Duplicates QA/QC

Midas Gold generated 1,778 quarter core duplicates from core holes of which 1,055 were above 0.025 ppm by gold fire assay and 102 were above 0.05% antimony. Reproducibility for quarter core duplicates was fair for both gold and antimony with a RMS CV of 21.1% for gold and 29.7% for antimony however the correlation coefficients for both are excellent at 0.94 (i.e. 1 is perfect). In addition, removal of outliers significantly improves the RMS CV.

Midas Gold generated a total of 536 RC field rejects of which 365 were above 0.025 ppm by gold fire assay, and 19 were above 0.05% antimony. Reproducibility for RC field rejects was poor to fair for both gold and antimony with an RMS CV of 23.5% for gold and 18.8% for antimony, respectfully. Figure 11.4 shows a scatter plot of both field duplicate types. The correlation coefficient for the gold trendline is 0.88 and warrants investigation in the future and should be considered for planning of future drill programs. As there are only 19 data points with which to compare for the antimony and the removal of one outlier bringing the correlation coefficient from 0.33 to 0.97, it appears that there is good agreement. Future anomalous values should be re-assayed.

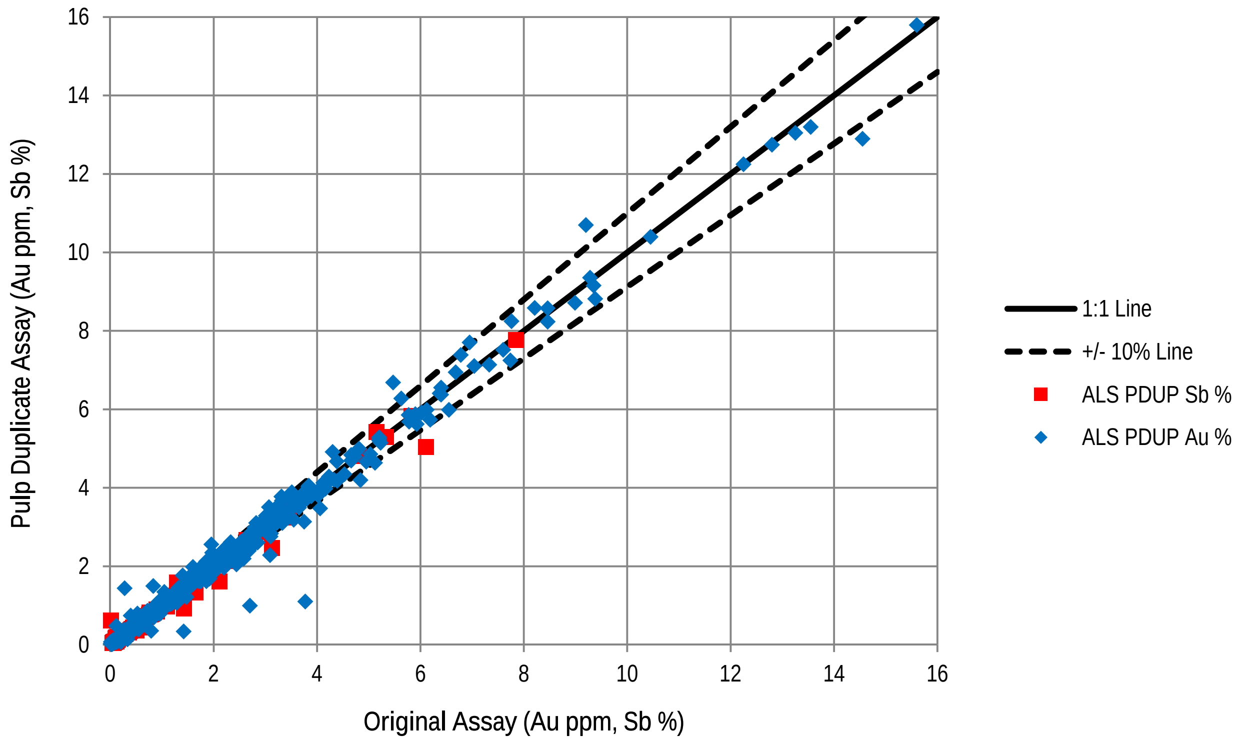
Figure 11.4: Field Duplicates



11.7.7 Pulp Duplicates QA/QC

ALS prepared one pulp duplicate for every twenty samples submitted. A total of 2,896 pulp duplicates were produced and assayed of which 1,646 were above 0.025 ppm for gold and 143 were above 0.05% antimony. Reproducibility for pulp duplicates was excellent for gold with an RMS CV of 6.8% and reproducibility was good to moderate for antimony with an RMS CV of 10.3%. Figure 11.5 shows scatter plots of the original assay values versus the pulp duplicate values.

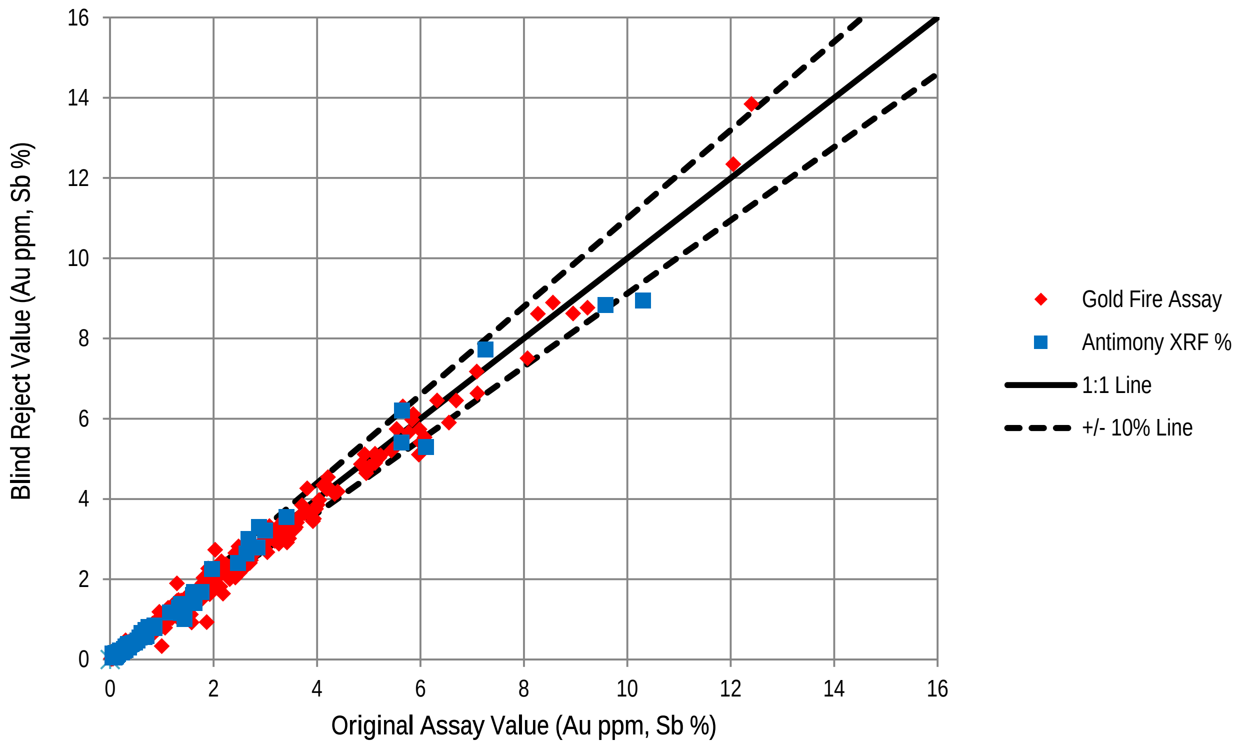
Figure 11.5: ALS Pulp Duplicates



11.7.8 Check Assays QA/QC

Midas Gold re-submitted 822 rejects with new sample numbers to ALS for assay to test for reproducibility and consistency (blind rejects). Out of the submitted rejects, 757 were above 0.025 ppm by gold fire assay and 99 were above 0.05% antimony by x-ray fluorescence (XRF). Within these parameters, the RMS CV for gold was 5.5% and the RMS CV for antimony was 9.4%, both values showing good reproducibility. A scatterplot of these values is shown on Figure 11.6.

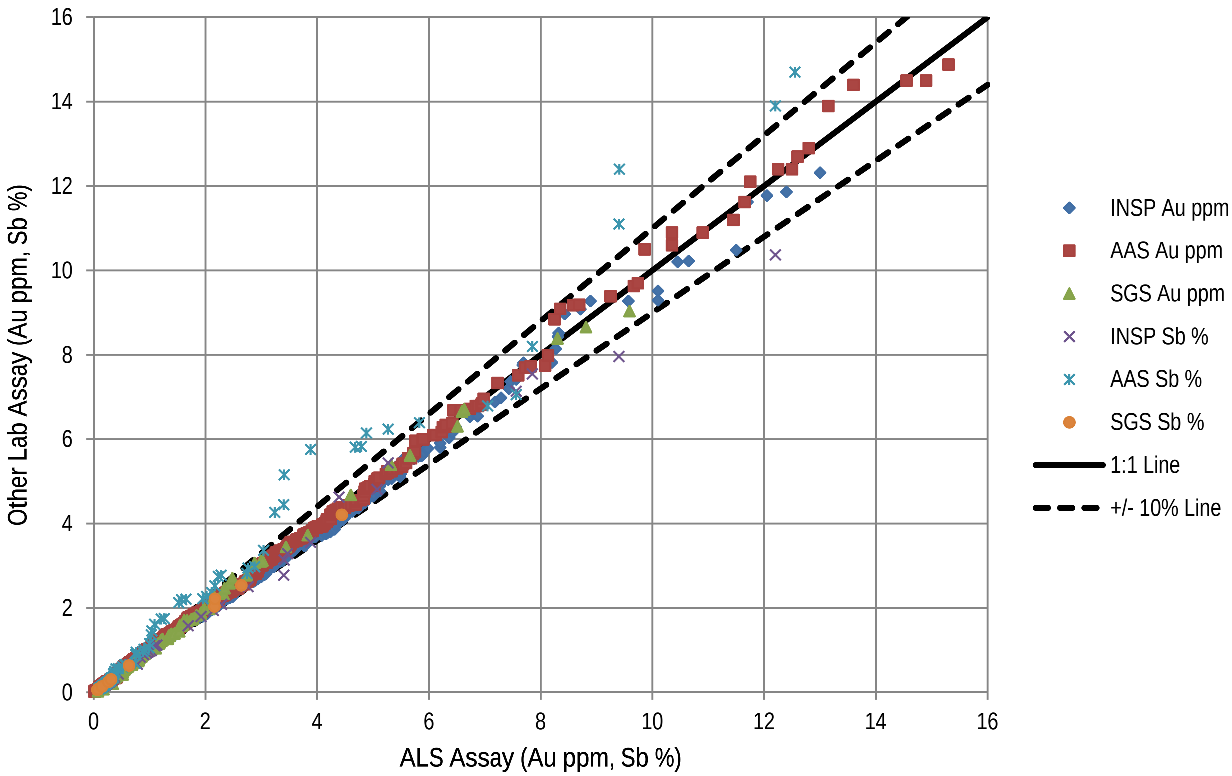
Figure 11.6: Blind Rejects Assays



Pulps were submitted to three different ISO certified laboratories for umpire assays as a cross check of ALS performance including: American Assay Labs, Inspectorate, and SGS. A total of 988 pulps were submitted to Inspectorate for gold fire assay of which 957 were above 0.025 ppm. The average percent difference between the Inspectorate assay and the reported ALS assay was -4.73%. Of these samples, 97 were also assayed for antimony of which 51 exceed 0.05% antimony. The average percent difference between ALS and Inspectorate antimony assays for these samples was -5.77%. A total of 1,003 pulps were submitted to AAS for gold fire assay of which 904 were above 0.025 ppm and 80 samples were assayed for antimony that exceeded 0.05%. The average percent difference between the AAS assay and the reported ALS assay was 4.44% for gold and 22.96% for antimony. Removal of samples outliers reduces the average difference to 9%. It appears that there may have been sample numbering issues or possibly poor assay methods by the lab itself.

SGS analyzed 92 samples of which 53 were assayed for gold only and 39 were assayed for gold and antimony. Ninety-one samples were above 0.025 ppm gold and 13 samples were above 0.05% antimony. The average percent difference between the SGS assay and the reported ALS assay for gold was 1.49% and for antimony was -0.08%. Figure 11.7 shows the QQ plot of umpire laboratory check assays of pulps.

Figure 11.7: QQ Plot of Pulp Check Assays



11.7.9 Work Order Evaluation and Corrective Actions

Assay shipments containing samples, duplicates, standards and blanks are grouped as work orders. Beginning in 2012, each standard and blank within ALS work orders was systematically evaluated using the criteria discussed in Sections 11.7.4 and 11.7.5. Work orders for 2011 were retroactively evaluated. Upon evaluation, several work orders contained standards or blanks which failed. Failed standards or blanks were re-assayed along with the 5 samples sequentially above and below the failure. Some work orders required assay revisions and others contained results that were confirmed by re-assay. When necessary, ALS would re-issue revised certificates and the Midas Gold database was updated accordingly. Table 11.6 summarizes the total and revised work orders by year.

Table 11.6: Work Orders and Revisions by Year

Year	Work Orders	Flagged Work Orders	Work Orders with Original Results Confirmed	Revised Work Orders
2011	189	27	23	4
2012	325	52	42	10
2013	82	6	3	3

11.8 CONCLUSIONS

Sample collection, preparation, analysis and security for all Midas Gold drilling are consistent with appropriate methods for disseminated gold–antimony–silver deposits:

- Midas Gold drill programs included insertion of blank, duplicate and standard reference material samples;
- Midas Gold QA/QC program results do not indicate any problems with the analytical programs or procedures;
- Midas Gold data are subject to validation, which includes checks on lithology data, mineralization/alteration data, sample numbers, and assay data. The checks are appropriate and consistent with industry standards;
- independent data audits have been conducted, and indicate that the sample collection and database entry procedures are acceptable; and
- all core has been catalogued and stored in secure designated areas and is appropriately safeguarded against weather.

Where historical data are available, sample collection, preparation, analysis, and security for pre-Midas Gold drill programs, are generally considered to have used accurate methods for disseminated gold–antimony–silver deposits but can only be partially verified with appropriate supporting QA/QC results. The QP is of the opinion that the quality and reliability of the sample collection methods, sample security protocols, sample preparation and gold, antimony, and silver analytical data from the pre-Midas Gold drilling programs is sufficient to support their use in Mineral Resource and Mineral Reserve estimation with the exception of certain holes flagged and determined to be unreliable due to lack of supporting data, poor sample quality, lack of survey control, inappropriate analytical methods or reporting limits or obvious bias. This assumption of validity is based on various reviews including analysis and inspection of original drill logs, assay certificates, paired data analysis between pre-Midas Gold drilling and Midas Gold drilling, assessment of geological continuity between pre-Midas Gold and Midas Gold drill holes, density of drilling, available pre-Midas Gold operator laboratory check assays and standards and inter-hole continuity.

The QP is of the opinion that the quality of the gold, antimony, and silver analytical data from Midas Gold drill programs is sufficiently reliable to support their use in Mineral Resource and Mineral Reserve estimation with the exception of certain reverse circulation holes that are flagged for exclusion due to cyclicity issues.

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12 DATA VERIFICATION

12.1 INTRODUCTION

Data verification programs have been undertaken by numerous independent consultants and by Midas Gold personnel, as discussed in previous NI 43-101 technical reports (SRK, 2011; SRK, 2012) and performed subsequently. This section summarizes the verification work performed on data and practices for both historical and current data. The Independent Qualified Person (QP), Garth Kirkham, P. Geo., believes that the datasets used for the mineral resource estimates are validated and verified as adequate for the estimation of mineral resources for each of the respective deposits.

The QP visited the Lake Fork, Idaho offices and facilities on April 23 - 25, 2014 and subsequently visited the site, facilities and surrounding areas on July 13 - 16, 2014.

The tour of the offices, core logging and storage facilities showed a clean, well-organized, professional environment. Onsite staff led Kirkham through the chain of custody and methods used at each stage of the logging and sampling process. All methods and processes are to industry standards and best practices and no issues were identified.

Four complete drill holes were selected by Kirkham and laid out at the core storage area. Site staff supplied the logs and assay sheets for verification against the core and the logged intervals. The data correlated with the physical core and no issues were identified. In addition, Kirkham toured the complete core storage facilities. No issues were identified and recoveries appeared to be very good.

The site visit entailed inspection of the shops, offices, reclaimed drill sites, the Yellow Pine, Hanger Flats and West End mineral resource areas along with the outcrops, historic drill collars and areas of potential disturbance for potential future mining operations. In addition, the site visit included a tour of the village of Yellow Pine, ID, which is the most likely populated area to be affected by any potential mining operation along with surrounding environs.

Kirkham is confident that the data and results are valid based on the site visit and inspection of all aspects of the Project, including methods and procedures used.

It is the opinion of Kirkham that all work, procedures, and results have adhered to best practices and industry standards required by NI 43-101. No duplicate samples were taken to verify assay results, but Kirkham is of the opinion that the work is being performed by a well-respected company and management that employs competent professionals that adhere to industry best practices and standards. Kirkham also notes that authors of prior technical reports (SRK, 2011; SRK, 2012) collected duplicate samples and had no issues.

12.2 MIDAS GOLD DATA REVIEWS

Midas Gold professional personnel have constructed and maintained the drill hole and geologic solids databases in-house since Project inception. A database geologist is supervised by an on-site resource geologist who is responsible and accountable for all of the data stored in the drill hole database and GEMCOM project directories. At intervals, Midas Gold has augmented, revised, and corrected its database in the following respects:

1. new drilling information from ongoing campaigns;
2. addition of cyanide soluble gold assays for West End drill holes completed by Midas Gold;
3. addition of QA/QC from previous gaps and ongoing Midas Gold drilling;
4. pre-Midas Gold drilling collar coordinate revisions and minor changes to Midas Gold hole collars;
5. corrections to Barrick Gold Corporation (Barrick) down hole surveys;

6. addition of lithology codes for certain pre-Midas Gold drill holes;
7. revision of below-detection assay value assignments;
8. assay precision changes;
9. naming of hole pre-collars and elimination of duplicates; and
10. numerous minor changes and additions to database tables and structures.

Midas Gold and its contractors have conducted numerous audits of manual inputs of pre-Midas Gold drill hole information from original paper log copies. In-house audits completed by Midas Gold geologists include a 100% audit of drill hole collar locations (March, 2013), a 5% audit of pre-Midas Gold assay records (January, 2013), a 100% audit of gold assays and lithology records for the West End Deposit (April, 2013) and a 100% audit of USBM assay records for the Yellow Pine Deposit. In addition, Midas Gold routinely electronically verifies assay records in the drill hole database against original electronic laboratory certificates. Independent contractors completed a 1% audit of pre-Midas Gold assay records against the original paper log copies and a 5% audit of Midas Gold assay records against PDF lab certificates (February, 2014) and a 100% electronic audit of Midas Gold Yellow Pine assay records against original electronic lab certificates.

12.3 PRE-MIDAS GOLD DRILL HOLE DATA

Historical drill holes on the Stibnite Gold property comprise 58% of the drill hole database by length and utilized a range of drilling, sampling and assaying methods over an 80 year period. Some historical drill data sets are characterized by factors which may impact the accuracy of the assay results including small diameter core, poor core recovery, assaying of sludge or sludge + core, reverse circulation or rotary drilling methods, disparate assay and analytical methods, and other factors. Midas Gold and its contractors have completed numerous projects to assess the accuracy of the historic drill hole data and evaluate what data sets are appropriate for estimation of mineral resources.

Midas Gold and previous operators on the property have conducted extensive confirmation drilling programs which provide the basis for statistical and graphical comparisons. Generally, confirmation drilling has tested areas previously drilled by the historical operators, but twinning of specific holes has been limited due to logistical restrictions based on post-mining topography and lack of access to historical underground development workings. Results of the statistical comparisons are typically reviewed using quantile-quantile plots, histograms and descriptive statistics.

Paired sample analysis has been employed at all three bedrock deposits to assess population characteristics of assay data for different drill programs. The analysis selects samples or composites from two drill data sets within a specified radius of one another (typically 10 to 30 m) and compares either: (a) all possible sample pairings; or (b) only the nearest unique sample pairings. The methodology comparing all sample pairings accentuates any clustering in the input data. A comparison of nearby samples selects sample groups representative of similar regions of the deposit by flagging all data points within a specified radius of samples in the comparative data set. Unlike paired sample analysis for all possible pairings, the populations for comparison can have different numbers of data points.

Nearest neighbor cell declustering utilizes a nearest neighbor estimation to assign grades to small blocks (typically 3 to 5 m cubes) within a specified anisotropic search radius from different sample or composite data sets. Blocks receiving an estimate from both data sets are compared statistically.

In an effort to achieve a more realistic cell size, data comparisons within drill “panels” takes the length weighted average grade of samples occurring within rectangular blocks or panels (typically 42 x 42 x 6 m) aligned approximately parallel to mineralization and orthogonal to drill holes.

Kriged blocks in common use standard linear geostatistical estimation methods to estimate mineral resource model block grades using the different drill data sets. Blocks estimated from each dataset are compared statistically.

12.3.1 Hangar Flats

Data evaluation at Hangar Flats focused on the pre-1955 historical drill holes consisting of three campaigns which comprise approximately 22% of the overall Hangar Flats database. These include the 1929 BMC-series underground core holes, circa 1950s MC-series underground and surficial core holes drilled by the Bradley, and circa 1950s DMA-series underground holes drilled by the USBM. These historical drill holes were BX-EX diameter, have core recoveries <50% and assayed sludge only.

12.3.1.1 Comparison of Pre-1955 Drilling Campaigns

The 1929 BMC-series underground drill holes at Hangar Flats are excluded from the mineral resource estimation process due to insufficient supporting information, poor core recoveries and uncertain assay methodologies of the period. The 1950s DMA series holes were reviewed graphically relative to Midas Gold drill holes and selected holes were excluded if assays were not corroborated by Midas Gold drilling or if assays did not adequately define the limits of the mineralized zones. Some MC-series drill holes were excluded from the mineral resource estimation dataset due to incomplete sample data. The remaining 38 MC-series and 22 DMA-series holes were evaluated relative to the combined Midas Gold and Hecla assay data using paired sample analysis methods, as summarized in Table 12.1. Results for gold indicate good agreement between data sets within 5 m but substantial high bias for pairs within 10 m. High bias in the historical underground holes is attributed to different orientations of the drill holes, with many Midas Gold drill holes drilled at a low angle to the MCFZ, while underground holes were drilled in more favorable orientations across relatively narrow high-grade zones in the underground development workings.

Table 12.1: Summary of Pre-1955 Drilling Campaign Evaluations for Hangar Flats

Method	Search Radius (m)	# Pairs		# Holes		Mean Gold Grade (g/t)		Mean Antimony Grade (%)	
		Gold	Antimony	Modern	Historic	Modern	Historic	Modern	Historic
Paired samples (nearest)	5	96	-	16	14	1.61	1.69	-	-
Paired samples (nearest)	10	376	167	31	25	1.22	1.62	0.08	0.34
Paired samples (all)	10	3,653	-	35	31	0.97	1.53	-	-

Paired sample analysis for antimony indicates substantial high bias in the historic data sets, which is potentially also attributed to drill hole orientation. Because they are more favorably oriented to the steeply dipping mineralization in the MCFZ than the Midas Gold and Hecla data, and also because they comprise only 11% of the data set at Hangar Flats, the pre-1955 MC- and DMA-series drill holes are retained for the purposes of mineral resource estimation for gold and antimony but are subjected to a strategy which limits their influence, effectively mitigating the impact of the potential assay bias with tighter spatial constraints on mineralization.

12.3.2 West End

The West End database contains 889 historical drill holes completed by a variety of operators using numerous drilling methods. Of these, 260 are air-track and rotary holes that are omitted from the dataset used for resource estimation. The remaining historical holes consist primarily of RC holes drilled by Pioneer and Superior, and core or RC holes drilled by Superior. Drill holes were evaluated using the paired sample analysis and blocks-in-common methods with results summarized in Table 12.2. With the exception of one analysis with very few samples, all campaigns yield means within +/- 10%, demonstrating that the fire-assay data for the historical campaigns is unbiased relative to the Midas Gold data and is suitable for the purposes of resource estimation.

Table 12.2: Summary of Post-1973 Drilling Campaign Evaluations for West End

Method	Campaigns	Search Radius (m)	# Pairs Gold	# Holes		Mean Gold Grade (g/t)	
				Modern	Historic	Modern	Historic
Paired sample analysis	Midas Gold vs Superior	10	497	2	5	1.76	1.91
Paired sample analysis	Midas Gold vs Pioneer	10	4,543	27	25	0.56	0.53
Paired sample analysis	Midas Gold vs Superior	10	602	7	7	1.07	0.67
Kriged blocks in common	Midas Gold vs Pioneer	40	27,968	-	-	0.625	0.562
Kriged blocks in common	Midas Gold vs Superior	40	19,568	-	-	0.731	0.693

12.3.3 Yellow Pine

The historic Yellow Pine drill hole database contains 768 historical drill holes which can be broadly divided into pre-1953 small diameter core holes and post-1973 core and RC drill holes.

12.3.3.1 Comparison of Post-1973 Drilling Campaigns

The post-1973 drill holes within the central region of the Yellow Pine Deposit were drilled by Ranchers and Barrick. Within the Homestake area of the Yellow Pine Deposit, north of 4,976,600 m, historical data sets consist primarily of drilling by Hecla and Ranchers, with a small number of Superior drill holes. Comparison results for gold are summarized in Table 12.3 and generally indicate good agreement of both Ranchers and Barrick gold assays with Midas Gold data. Ranchers antimony assays compare well to Midas Gold but Barrick antimony appears to be low-biased, presumably due to assaying of antimony on longer 12.19 m (20 ft) intervals. A comparison of the Hecla 1980s RC drill holes with Midas Gold data illustrates varying degrees of high-bias in the Hecla gold data set. To further evaluate this difference, Hecla data was compared to Ranchers data as the drilling covers a similar area and the Ranchers data shows good agreement with the Midas Gold data. These comparisons indicate a persistent high-bias in the Hecla data with respect to both Midas Gold and Ranchers data.

To quantify the potential impact of high-biased Hecla data on the mineral resource estimate, a model sensitivity analysis was run for the Homestake domain only and indicated a ~3% increase in contained gold ounces (~11,000 oz Au) with inclusion of the Hecla data versus the Hecla data removed. It was decided to retain the Hecla data in the final dataset for mineral resource estimation because the change in contained metal is not significant, and because removal of the data would force the model to extrapolate grade across greater distances.

Table 12.3: Summary of Post-1973 Drilling Campaign Evaluations for Yellow Pine

Campaigns	Method	Search Radius (m)	# Pairs Gold	Mean Gold Grade (g/t)	
				Midas Gold	Historic
Midas Gold vs Post-1973 (all)	Paired samples	5	251	2.08	2.32
Midas Gold vs Post-1973 (Central Yellow Pine)	Nearby samples	20	566 / 798	2.55	2.11
	Nearby samples	40	1,526 / 2,050	2.26	2.32
Midas Gold vs Barrick (Central Yellow Pine)	Nearest neighbor declustering	20	16,573	1.83	1.78
	Drill panel comparison	42 x 42 x 6	92	2.26	2.49
Midas Gold vs Ranchers (Central Yellow Pine)	Nearest neighbor declustering	20	14,732	2.40	2.41
	Drill panel comparison	42 x 42 x 6	83	2.45	2.39
Midas Gold vs Ranchers (Homestake area)	Nearby samples	20	369 / 185	1.38	1.12
	Nearby samples	40	551 / 298	1.16	1.05
	Nearest neighbor declustering	20	5,306	1.34	1.27
	Drill panel comparison	20	16	1.00	1.13
Midas Gold vs Hecla (Homestake area)	Nearby samples	20	459 / 271	1.45	1.41
	Nearby samples	40	760 / 447	1.41	1.57
	Nearest neighbor declustering	20	7,930	1.12	1.50
	Drill panel comparison	42 x 42 x 6	34	1.36	2.19
Ranchers vs Hecla (Homestake area)	Declustered mean grade	N/A	569 / 688	1.17	1.38
	Kriged blocks in common	60 x 45 x 25	3,259	1.19	1.52

12.3.3.2 Comparison of Pre-1953 Drilling Campaigns

The pre-1953 drill holes at Yellow Pine consist of Bradley and USBM drill holes, primarily drilled in the 1940s. Both the Bradley and USBM drill holes are commonly characterized by poor core recoveries associated with small diameter core and drilling technology utilized at the time, unverifiable surveyed drill hole positions, absence of a documented quality control program and sampling of sludge only (Bradley) or sludge + core (USBM). For the USBM drill holes, the Midas Gold database contains the weighted average grade based on the dry weight of core and sludge as calculated by the USBM and preserved on historic log-sheets. The averaging method applies a weighted average grade, but does not incorporate the theoretical recovery based on the hole diameter and drill bit annulus, as was standard practice at the time. Midas Gold, Barrick and Ranchers drill hole data show good agreement for gold, and Midas Gold and Ranchers show good agreement for antimony and these data sets are respectively utilized for evaluation of the these metals in the pre-1953 drill holes.

A total of 187 pre-1953 drill holes were flagged by Midas Gold and removed from the dataset including the 1920s Bradley churn drill holes, holes missing critical collar or assay data, and holes for which positions could not be verified on historic maps and cross sections. The remaining 179 pre-1953 drill holes were statistically evaluated on an overall basis, within specific regions of the deposit, and within sub-groups based on period and drill hole series, and surface versus underground collar positions.

The USBM drill holes consist of 52 surficial holes drilled in 1939 and 1940. As summarized in Table 12.4, the USBM drilling campaigns generally compare well to Midas Gold and post-1973 campaigns for both gold and antimony. For the central region of Yellow Pine, comparison methods indicate historic assays are within +/- 10% of post-1973 data. Within the southern region of the Yellow Pine Deposit, comparisons indicate both high- and low-bias using different methods, which is attributed to the limited number of samples and spatial bias rather than any persistent analytical bias. USBM drill holes were therefore retained in the dataset and used for the purposes of mineral resource estimation.

Table 12.4: Summary of Pre-1953 Drilling Campaign Evaluations for Yellow Pine

Campaigns	Comparisons	Method	Search Radius (m)	# Pairs Gold	Mean Gold Grade (g/t)	
					Midas Gold	Historic
USBM	Modern vs USBM (All)	Paired samples	5	61	1.65	1.61
		Nearby samples	40	2,808 / 866	2.35	2.12
	Modern vs USBM (YP South)	Nearby samples	20	33 / 62	2.59	0.84
		Nearby samples	40	58 / 152	2.54	0.78
	Midas Gold vs USBM (YP South)	Nearest neighbor declustering	20	3,598	0.63	1.40
		Drill panel comparison	42 x 42 x 6	6	0.79	1.03
	Midas Gold vs USBM (YP Central)	Nearest neighbor declustering	20	12,643	2.15	2.30
		Drill panel comparison	42 x 42 x 6	49	2.42	2.53
	Modern vs USBM (YP Central)	Nearby samples	20	916 / 737	2.34	2.37
		Nearby samples	40	2,106 / 1,132	2.33	2.34
Bradley Drill Holes	Modern vs Bradley (All)	Paired samples	5	125	2.01	2.76
	Modern vs B-series (All)	Nearby samples	20	1,338 / 462	2.47	2.52
	Modern vs B-series (YP Central)	Nearby samples	20	1,289 / 410	2.48	2.65
		Nearby samples	40	2,724 / 746	2.37	2.30
	Midas Gold vs Bradley (YP Central)	Nearest neighbor declustering	20	21,753	2.45	2.77
	Midas Gold vs Bradley (YP South)	Nearest neighbor declustering	20	1,438	1.59	1.65
		Drill panel comparison	42 x 42 x 6	3	1.94	1.97
	Midas Gold vs Bradley (Homestake)	Nearest neighbor declustering	20	3,709	1.12	1.8
	Midas Gold & Barrick vs Bradley 1940s surf. (YP Central)	Nearest neighbor declustering	20	33,361	2.50	2.57
		Drill panel comparison	42 x 42 x 6	169	2.55	2.84
	Midas Gold & Barrick vs Bradley 1950s surf. (YP Central)	Nearest neighbor declustering	20	6,166	2.27	1.95
		Drill panel comparison	42 x 42 x 6	12	2.20	2.00
Bradley Underground Drill Holes	Modern vs T-series UG (YP Central)	Nearby samples	20	824 / 899	2.74	2.94
		Nearby samples	40	1,802 / 1,371	2.25	2.74
	Midas Gold & Barrick vs Bradley 1940s U.G. (YP Central)	Nearest neighbor declustering	20	17,404	2.58	2.77
		Drill panel comparison	42 x 42 x 6	44	2.47	2.59
	Midas Gold & Barrick vs Bradley 1950s U.G. (YP Central)	Nearest neighbor declustering	20	7,508	2.39	2.92
		Drill panel comparison	42 x 42 x 6	16	2.83	3.16
	Bradley Surf. vs Bradley U.G. (YP Central)	Nearby samples	10	64 / 97	3.27	3.39
		Nearby samples	20	170 / 296	2.19	2.71

The Bradley drill holes can be subdivided into the T-series (underground) and B-series (surficial and underground), and into those drilled in 1940 versus those drilled in the late 1940s and early 1950s. When compared to all Bradley holes not omitted previously, the post-1973 data is low-biased by between 5% and 40% within the central and southern regions of the deposit for gold while the Bradley drill holes are substantially high-biased in the Homestake (northern) region. A comparison of only the surficial holes reduces the apparent assay bias in the Bradley data considerably and some comparison methods indicate a low bias relative to the modern data sets. The underground drill holes consistently show good agreement with modern data at shorter distances with an increasingly positive bias at larger comparative distances. This same relationship is demonstrated relative to the Bradley surficial drill holes, which is attributed to the location of these holes being within the higher-grade regions of the deposit rather than to any persistent analytical or sampling bias in the underground drill holes.

For antimony, the Bradley drill holes show a consistently high-bias relative to Midas Gold and Ranchers drill holes using both statistical methods and graphical comparison. This is attributed to assay tailing within sludges below mineralized zones, which is much more significant with antimony than with gold.

Based on the results discussed above, the surficial Bradley and USBM holes were retained for use in the gold mineral resource estimate within the central and southern regions of the deposit; however, the Bradley drill holes in the Homestake region were removed. The underground drill holes were also retained, but their range of influence was restricted to 12 m in the gold mineral resource estimate. In addition, the Bradley drill holes are not utilized in the antimony estimate at all. To quantify the potential impact of pre-1953 data on the mineral resource estimate, model sensitivities were run with various combinations of historic data. The sensitivity incorporating the final data set used for the purposes of mineral resource estimation indicates a 4% increase in total contained gold when compared to using only the post-1973 data, which is well within acceptable limits.

12.3.4 Historic Tailings

The Historic Tailings database contains 25 historic auger drill holes drilled by Stibnite Mines Inc. (SMI) in the 1990s, in addition to drilling completed by Midas Gold. The historical holes have cyanide assays only and were not utilized in the mineral resource estimate.

12.4 CONCLUSIONS

Kirkham visited the Lake Fork, ID offices and facilities on April 23 - 25, 2014 and subsequently visited the site, facilities and surrounding areas on July 13 - 16, 2014. During these visits, no issues were identified and all procedures and protocols were to industry standards as expected for a North American operation at the pre-feasibility stage of development.

The datasets employed for use in the mineral resource estimates are a mix of historic data and current, modern data. There is always a concern with respect to validity of the historic data and extensive validation verification must be performed in order to insure that the historic data may be relied upon.

Kirkham reviewed extensive validation and verification procedures and results performed by external consultants and by Midas Gold in order to ensure validity of the Mineral resource estimates and for classification purposes. The methods and procedures performed by Midas Gold were carried-out with great care, and were supervised and approved by Kirkham, which entailed detailed analysis and resulted in sub-sets of data being excluded or, in some cases, being flagged so as reduce their influence due to any potential bias.

It is the opinion of Kirkham that the data used for estimating the Mineral Resources for the Hanger Flats, West End, Yellow Pine and Historic Tailings deposits is adequate for this purpose and may be relied upon to report the Mineral Resources and Mineral Reserves contained in this Report.

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13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Metallurgical testing has been conducted on samples from the Yellow Pine, Hangar Flats, West End and the Historic Tailings deposits. This work has included extensive mineralogical studies and developmental metallurgical test work on various ore types from each of the deposits. Despite the differences in the deposits, developmental metallurgical testwork has been able to identify a flowsheet that proved successful when applied to each of the deposits, making it possible to design a single plant that can process all ores from the Project as they are mined.

An auriferous pyrite recovery flotation circuit was developed to process the low antimony sulfide ores of all three deposits. This circuit consisted of roughers and a single stage of cleaning with scavenging, with scavenger tailings operating with the option to flow closed circuit back to the primary ball mill (preferred option). At times, the rougher concentrates may meet the pressure oxidation (POX) requirements and not require further cleaning.

When antimony-rich sulfide ores are being processed, a smaller antimony recovery circuit would be operated ahead of pyrite flotation. To produce saleable stibnite concentrate, this circuit requires roughing followed by two stages of cleaning, with the antimony 1st cleaner tailings recombining with the rougher tailings to feed the pyrite circuit. The antimony 2nd cleaner tailings are returned to the 1st cleaner feed.

Additionally, test work was initiated to assess the ability to reject carbonate-bearing (CO_3) minerals from the gold concentrates of the carbonate-rich West End Deposit, which interfere with the ability to pressure oxidize the concentrates without pre-acidulation. West End sulfide ores can successfully produce concentrate with a POX friendly carbonate to sulfur ratio through one stage of cleaning; to maintain gold recovery, a scavenger and recirculation of scavenger tailings to the primary mill for reprocessing is recommended.

Developmental leaching test work was also undertaken on the West End oxide ores as well as on select flotation tailings produced from partially oxidized mineralization from Hangar Flats and West End. West End oxide leach studies indicate that 96% of the extracted gold leaches in the first six hours. Leach studies on the flotation tailings from Hangar Flats and West End indicate that gold in the flotation tailings are also fast leaching and could contribute substantially to gold recovery.

Below can be found a description of the grindability of the materials and the mineralogy of the deposits. Following this, for each deposit, a brief reference to past operations and testwork on ores and samples from each deposit is made followed by a summary of the metallurgical test data.

13.2 SAMPLE SELECTION AND COMPOSITE PREPARATION

Approximately 800 core samples from the Yellow Pine, Hangar Flats and West End deposits were delivered from the site to SGS Vancouver for mineralogical and metallurgical studies during 2013 and 2014. Sonic and auger samples from the Historic Tailings Deposit and geochemical laboratory assay reject samples from the West End Deposit were also tested. The sources of the samples from the deposits are shown on Figure 13.1, Figure 13.2 and Figure 13.3.

Figure 13.1: Sources of Samples for Yellow Pine and West End Metallurgical Testing

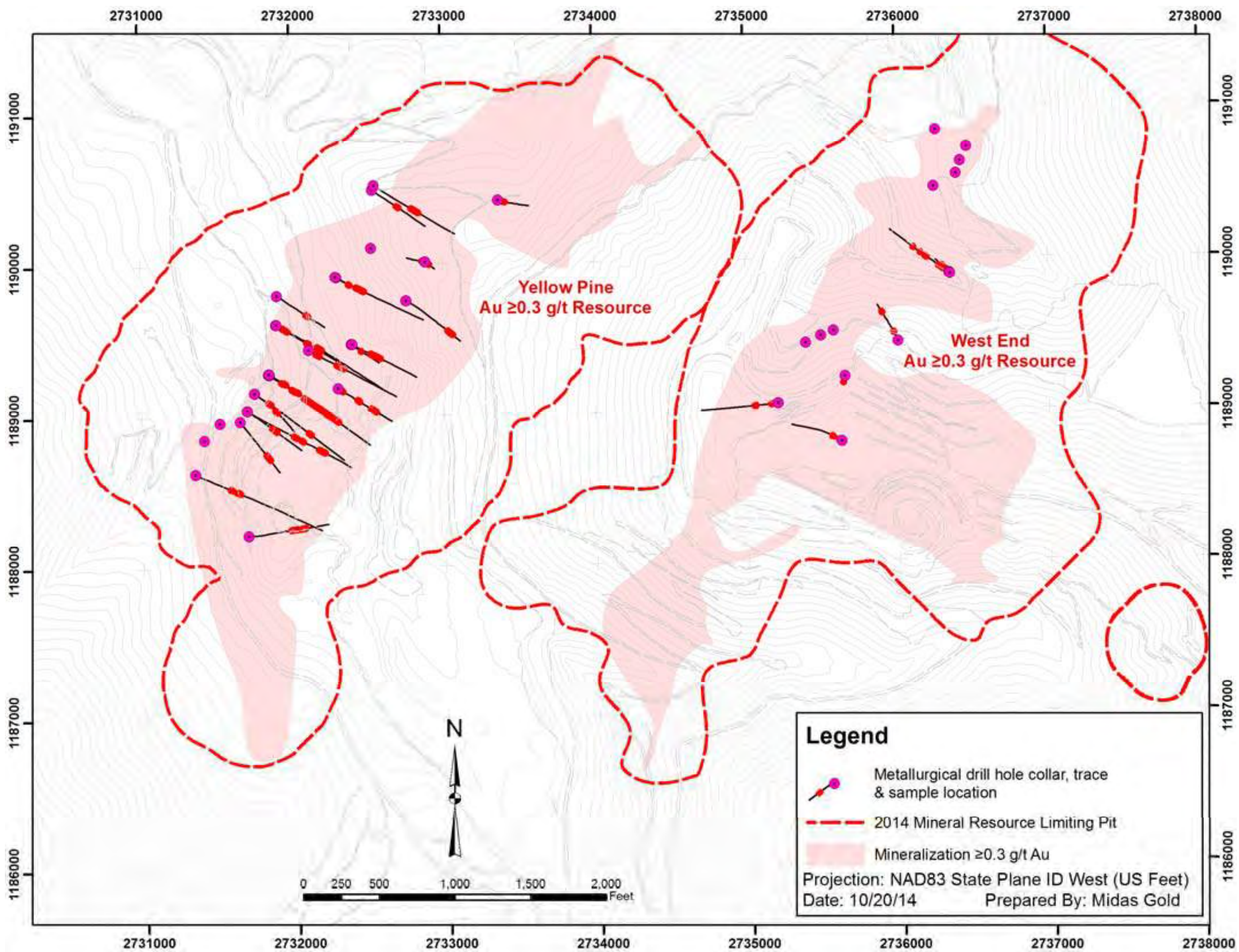


Figure 13.2: Sources of Samples for Hangar Flats Metallurgical Testing

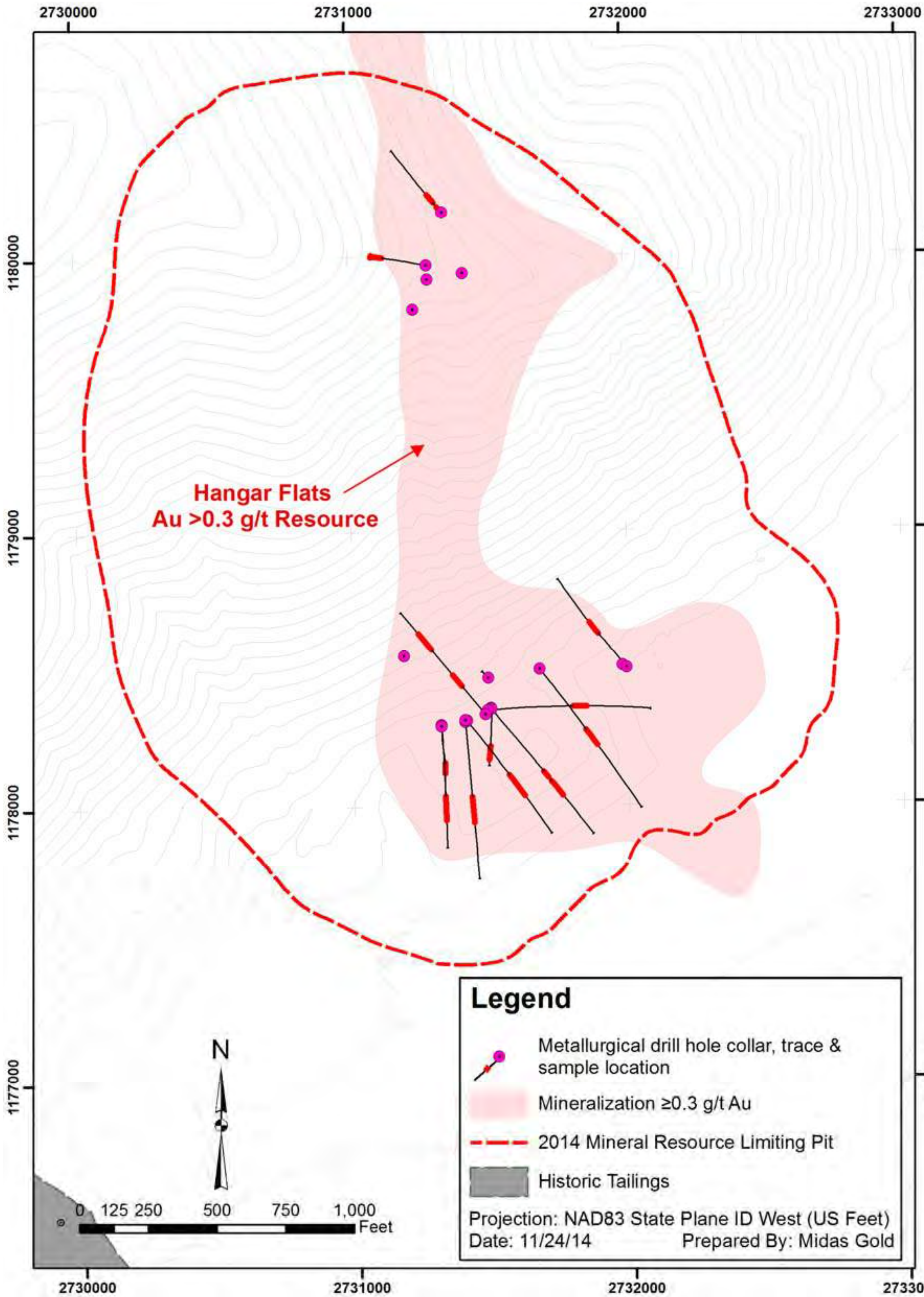
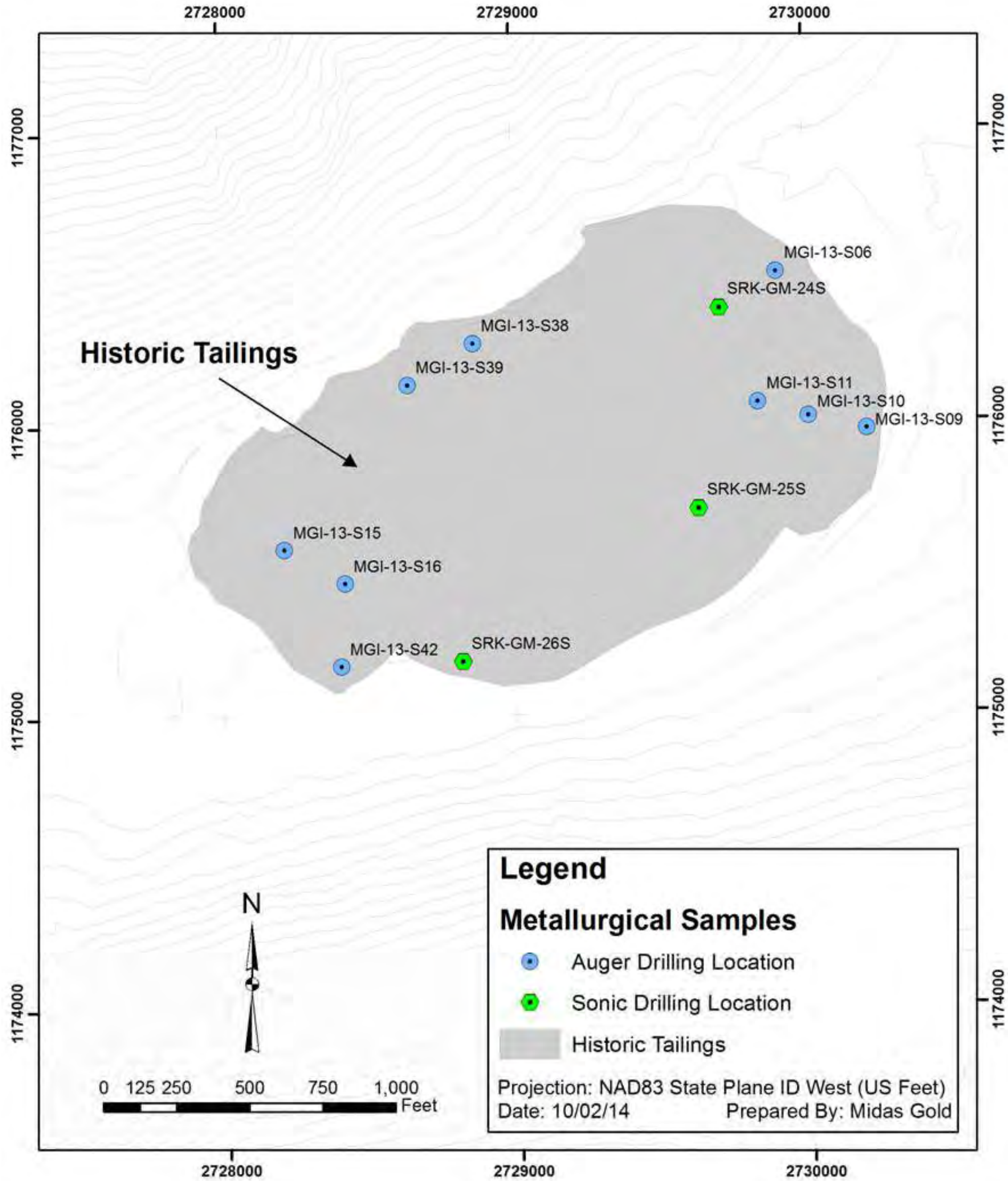


Figure 13.3: Sources of Samples for Historic Tailings Metallurgical Testing



From these samples, a variety of composites were created for various testing and characterization purposes (Table 13.1). A few composites were used that were remaining from the PEA program, as were some of the concentrate products tested. In addition, some 114 variability composites were created, including 43 from Yellow Pine, 11 from the Yellow Pine early production zone, 27 from Hangar Flats, 12 from the Historic Tailings Deposit and 16 from West End, in part to assess the variability in bulk mineralogy across the deposits.

Table 13.1: Primary Metallurgical Composites for Testing

Comp. ID	Description	No. of Tests	Purpose	Head Grades					
				Au (g/t)	Ag (g/t)	As (%)	Sb (%)	$S_{(t)}$ (%)	W (%)
SubA1	HF Global	12	Used PEA concentrate for testing	3.49	6.3	0.690	0.190	1.34	n/a
SubA2	WE Global	12	Used PEA concentrate for testing	2.05	2.5	0.330	0.010	0.89	n/a
SubA3a	YP High Sb	3	Used PEA concentrate for testing	2.87	1.5	0.450	0.190	1.47	n/a
SubA3b	YP Low Sb	9	Used PEA concentrate for testing	2.18	2.2	0.370	0.010	0.97	n/a
I	HF high tungsten	7	PEA Comp - Scoping tungsten recovery	2.31	185.0	0.180	13.00	6.06	1.600
YP104	YP	7	Scoping tungsten recovery	1.51	15.4	0.180	0.820	1.41	0.420
P2	Dup Comp P (YP)	16	PEA comps for production of Au concentrate	2.25	n/a	0.370	0.000	1.08	n/a
YPH	YP High Sb	39	Flowsheet development, locked cycle tests	2.14	4.1	0.244	0.380	1.03	n/a
YPL	YP Low Sb	25	Flowsheet development, locked cycle tests	1.82	2.8	0.353	0.043	0.99	n/a
HFH	HF High Sb	42	Flowsheet development, locked cycle tests	1.96	8.6	0.390	0.500	1.03	n/a
HFL	HF Low Sb	26	Flowsheet development, locked cycle tests	1.60	2.3	0.460	0.035	0.81	n/a
WES	WE Sulfide	23	Flowsheet development, locked cycle tests	1.79	1.7	0.218	0.027	0.65	n/a
WEO	WE Oxide	11	Oxide leach development	0.91	1.3	n/a	n/a	0.04	n/a
S06	HT High Sb	4	Flotation and leach confirmation testing	1.44	<10	0.099	0.871	0.61	0.070
S24	HT Low Au	9	Scoping flotation and leach testwork	0.98	4.0	0.092	0.140	0.43	0.016
S25	HT Avg Au	9	Scoping flotation and leach testwork	1.12	3.0	0.150	0.160	0.36	0.029
S26	HT High Au	10	Scoping flotation and leach testwork	1.51	3.8	0.180	0.220	0.29	0.023
HTL	HT Low Au	4	Flotation and leach confirmation testing	0.78	<10	0.091	0.074	0.18	0.012
HTM	HT Avg Au	6	Flotation and leach confirmation testing	1.17	<10	0.150	0.230	0.37	0.033
HTH	HT High Au	6	Flotation and leach confirmation testing	1.31	<10	0.170	0.170	0.28	0.021
EP1	Early YP Low Sb	3	Flotation and leach confirmation testing	2.87	n/a	0.602	<0.01	1.36	n/a
EP2	Early YP Low Sb	3	Flotation and leach confirmation testing	0.90	n/a	0.518	<0.01	1.29	n/a
EP3	Early YP Low Sb	3	Flotation and leach confirmation testing	2.94	n/a	0.226	0.070	1.01	n/a
EP4	Early YP Low Sb	3	Flotation and leach confirmation testing	2.40	n/a	0.264	<0.01	0.91	n/a
EP5	Early YP High Sb	4	Flotation and leach confirmation testing	2.53	n/a	0.577	0.540	1.41	n/a
EP6	Early YP High Sb	3	Flotation and leach confirmation testing	2.97	n/a	0.588	0.180	1.66	n/a
EP7	Early YP High Sb	3	Flotation and leach confirmation testing	3.35	n/a	0.316	0.930	1.74	n/a
EPL	Early YP Low Sb	2	Confirmatory flotation, locked cycle tests	2.22	3.4	0.425	0.030	1.21	n/a
EPH	Early YP High Sb	2	Confirmatory flotation, locked cycle tests	3.09	n/a	0.462	0.650	1.71	n/a
EPLB	EPL-HTM Blend	4	Confirmatory flotation, locked cycle tests	2.14	3.7	0.364	0.060	1.05	n/a
EPHB	EPH-HTM Blend	3	Confirmatory flotation, locked cycle tests	2.95	4.6	0.407	0.530	1.45	n/a

Note: YP = Yellow Pine, HF = Hangar Flats, WE = West End, HT = Historic Tailings, EP = Yellow Pine Early Production Zone

13.3 GRINDING CHARACTERIZATION

A total of twenty two SMC, twenty four Bond Ball Mill Work Index, eight Bond Rod Mill Work Index, ten abrasion index and ten crusher work index tests were conducted on composites to support the PEA and PFS metallurgical programs, taken from samples around each of the deposits. All the work was conducted by SGS Lakefield and SGS Vancouver; the results from these tests are provided in Table 13.2. All three deposits have average grindability characteristics of which, Yellow Pine is most resistant to ball milling and West End is the least amenable to SAG milling (Ratnayake, 2013a; Gajo, 2014b).

Table 13.2: Grinding Characterization Samples

Test	Units	Yellow Pine			Hangar Flats			West End		
		No. of Tests	Avg.	St. Dev.	No. of Tests	Avg.	St. Dev.	No. of Tests	Avg.	St. Dev.
JK Drop Weight SAG Testing										
A x b	N/A	0	n/a	n/a	1	123.2	n/a	1	63.4	n/a
ta	N/A	0	n/a	n/a	1	1.5	n/a	1	0.37	n/a
SMC Testing										
A x b	N/A	9	87.5	17.5	7	150.4	54	6	50	18.3
ta	N/A	9	0.86	0.18	7	1.5	0.53	6	0.49	0.18
Crusher and Mill Index Testing										
Crusher WI	kWh/Mt	2	5.1	0.07	5	7.4	1.6	3	11.6	1.9
Abrasion Index	N/A	2	0.26	0.003	5	0.22	0.02	3	0.24	0.11
Bond Rod Mill WI	kWh/Mt	2	10.9	0.21	4	10.9	0.93	2	14.7	2.1
Bond Ball Mill WI	kWh/Mt	10	14.1	0.8	7	13.3	0.58	7	13.0	0.64

13.4 MINERALOGY

Process mineralogical studies were conducted by SGS Vancouver, Process Mineralogy Consultants, Surface Science Western and Actlabs under the guidance of Blue Coast Metallurgy.

Full gold deportment studies were conducted on twelve samples (four from each deposit), while 140 samples were subjected to bulk mineralogical analysis using QEMSCAN (62, 50 and 28 from Yellow Pine, Hangar Flats and West End, respectively).

The gold is predominantly refractory to direct cyanidation, being present in solid solution or colloidal form in the host pyrite and arsenopyrite minerals. Discrete gold is particularly rare in the Yellow Pine and Hangar Flats deposits, but somewhat more abundant in the West End Deposit. Any discrete gold occurrences are very fine, typically ranging from 1 to 10 microns (µm) in size.

The vast majority of the gold hosted within the three deposits occurs as solid-solution gold, atomically dispersed within the sulfides and it seems likely that only the materials where the sulfides have been completely destroyed (true-oxide materials) host no solid-solution gold at all. The mean grades of the gold hosting sulfides, as identified using laser-ablation ICP-MS are provided in Table 13.3.

Both pyrite and arsenopyrite are not stoichiometric. The pyrite is often strongly arsenian and the arsenopyrite commonly arsenic-deficient. Accordingly, whereas in many deposits of this type the gold is enriched in arsenopyrite, at this Project it occurs in all iron sulfides. Gold is, however, primarily enriched within porous pyrite, fine pyrite and arsenopyrite. The coarse crystalline sulfides contain relatively little gold.

Antimony occurs as stibnite, which is typically coarse-grained when occurring in higher-grade samples. At head grades above 0.1% antimony, the stibnite mean grain size is typically 15 - 25 microns. As the antimony grade drops, the respective stibnite grain size drops markedly.

Table 13.3: Discrete and Solid Solution Gold Mineralogy

Gold Mineralogy		Yellow Pine	Hangar Flats	West End
Free-Milling Gold		1-5%	1-17%	5-86%
Refractory Gold Host		Grade of Gold in Host Mineral (ppm)		
Pyrite	Coarse	23	5	19
	Porous	42	168	216
	Disseminated	108	212	104
Arsenopyrite	Coarse	54	3	17
	Porous	62	77	152
	Disseminated	88	n/a	n/a
Stibnite		1	n/a	n/a

The host rock bulk mineralogy is shown in Table 13.4, which describes the median, 10th percentile and 90th percentile of each of the major components in a total of 140 samples analyzed by QEMSCAN to date. Some key features of these data include:

- Significant variability occurs in the modal mineralogy in each of the deposits;
- West End tends to be poorer in sulfides, so mass pull would be lower to the pre-oxidation circuit;
- West End has a hard quartzite component, which may be the cause of the harder grindability data;
- Clays are best represented in these data by the illite/muscovite category, and tend to be richest in the Hangar Flats Deposit; and
- Carbonates are richest in the West End Deposit.

Table 13.4: Distribution of QEMSCAN Modal Abundances

Deposit	Yellow Pine			Hangar Flats			West End		
Modal Abundance Percentile	10th	50th	90th	10th	50th	90th	10th	50th	90th
Pyrite/Arsenian Pyrite	1.1	2.2	3.0	0.6	1.8	3.0	0.3	0.8	2.2
Arsenopyrite	0.4	0.9	2.2	0.1	1.1	2.9	0.0	0.4	1.5
Gold-bearing Sulfides	1.7	3.1	5.1	1.1	3.1	5.2	0.4	1.2	3.4
Galena	0.0	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0
Stibnite	0.0	0.0	0.9	0.0	0.0	2.5	0.0	0.0	0.0
Quartz	28.4	35.2	43.0	30.3	36.6	47.3	17.8	32.5	73.4
Feldspar	31.4	41.8	51.0	23.8	36.5	46.7	3.5	16.9	40.2
Illite/Muscovite	5.8	10.9	18.3	7.8	16.0	23.4	5.5	12.3	27.4
Chlorite	0.0	0.0	0.1	0.0	0.1	0.2	0.1	0.2	0.9
Clays	0.0	0.4	1.5	0.7	1.4	2.0	0.5	1.4	4.9
Other Silicates	0.2	0.4	0.8	0.1	0.3	0.9	0.5	2.6	5.0
Oxides	0.3	0.6	1.0	0.2	0.4	1.1	0.3	0.7	1.6
Carbonates	1.9	3.5	7.2	0.1	1.7	4.6	2.5	13.2	41.1
Apatite	0.1	0.3	0.7	0.0	0.5	0.6	0.1	0.2	0.3
Other	0.1	0.2	1.1	0.1	0.7	2.0	0.0	0.1	3.7

13.5 YELLOW PINE DEPOSIT

13.5.1 Historical Metallurgy

The Yellow Pine Deposit has been mined intermittently since 1938. In the earlier years, flotation was employed to produce, during different historical periods, gold, antimony and tungsten concentrates, with milling rates reaching 2,200 short tons per day (st/d) prior to shutdown in 1951. In the late 1980s, a modest heap leach operation was commissioned to process Homestake oxide ores (located to the northeast of the main Yellow Pine Deposit). This was expanded with the construction of a new heap in 1990 with recoveries in the order of 80%. Heap leach operations phased down in 1991 and were discontinued in 1992 (Mitchell, 2000).

Several programs of testing have been conducted on Yellow Pine sulfide samples since the 1970s that included flotation of sulfide ores and pre-oxidation followed by cyanidation of the oxidized sulfide concentrates. Key programs were conducted at Hazen Research, Bacon, Donaldson and Sherritt Gordon in 1983 (Sherritt Gordon Mines Limited, 1983); at Lakefield Research, Mountain States R&D and the University of Idaho in 1987 (Rollwagen, 1987; Brackebusch, 1987) and at Lakefield Research and the University of Idaho in 1993 (Jackman, 1993; Harrington, Bartlett, & Prisbrey, 1993). Aside from conventional roasting and autoclaving, bio-oxidation was tested in 1987 and 1993 and various more novel processes in the 1980s.

13.5.2 Flotation

The focus of this Project is on maximizing gold recovery to a gold concentrate for auto-thermic pressure oxidation and doré production, while generating a saleable grade antimony concentrate (generally considered at least 50% Sb) from ores which contain economically significant antimony (assuming a feed cut-off of ~0.1% Sb).

The PEA results demonstrated that saleable grade antimony concentrates and oxidation-ready gold concentrates could be made from Yellow Pine's antimony-bearing ores ground to a product size of 80% passing (P_{80}) 100 µm. Using sodium cyanide to depress the pyrite and arsenopyrite in the antimony circuit, the optimum test yielded a 51% antimony concentrate at an antimony recovery of 75% with just 1% of the gold lost to the antimony concentrate. Subsequent gold flotation recovered roughly 90% of the gold to a rougher concentrate assaying 27 grams per metric tonne (g/t) gold (Au) and 12% sulfur (S).

For the prefeasibility program, flowsheet development was conducted on two composites; the Yellow Pine High Antimony (YPH) composite and the Yellow Pine Low Antimony (YPL) composite. These were each blended to represent the average feed grades of gold, antimony and sulfur for the antimony-rich material, and gold and sulfur for the antimony-poor material expected over the four phases of mining as set forth in the PEA. Confirmatory testing was also conducted on two major composites broadly representing key early production material (Early Production High antimony [EPH] and Low antimony [EPL]) and seventeen variability composites sourced from points around the Yellow Pine Deposit (Gajo, 2014a; Gajo, 2014b).

Most of the grind optimization testwork was completed on the low antimony composites, which represents material that is more plentiful in the deposit. This identified a grind product P_{80} of 75 µm to be the preferred target. This grind target was applied to the YPH composite ore and compared to the results of the same test at a target P_{80} of 100 µm. The finer grind was found to benefit the recovery of gold and sulfur in YPH while improving the selectivity against gangue minerals in both circuits. Reagent optimization testwork was then undertaken to:

1. On the YPH composite: produce concentrates of both stibnite (antimony) and pyrite/arsenopyrite (gold) while minimizing reagent use and maximizing recoveries within the respective circuits. The antimony flowsheet goals were to produce a concentrate grading 50% Sb while recovering a minimum of 70% of the antimony and rejecting the maximum amount of gold from the circuit. The gold flowsheet goals were to recover the maximum amount of gold to a concentrate initially grading 10% sulfur but later determined by

the autoclave design group to be sufficient at 5% sulfur for auto-thermic autoclaving. For the most part, gold rougher flotation achieved the 5% goal but cleaning flowsheets were also developed in the case that higher sulfur grades were needed for pre-oxidation. In the final flowsheet, ore was ground in a fully stainless steel environment to a P_{80} of 75 µm with 200 g/t lime and 75 g/t sodium cyanide; stibnite was floated using 355 g/t lead nitrate and 15 g/t Aerophine 3418A, then cleaned without regrinding using 10 g/t sodium cyanide in each of two stages of cleaning. Methyl isobutyl carbinol (MIBC) was used as the frother. The baseline flowsheet included subsequent pyrite/arsenopyrite rougher flotation, conducted using 400 g/t copper sulfate and 200 g/t potassium amyl xanthate (PAX).

2. On the YPL composite; grind optimization testwork was completed by testing the flotation response of gold at primary grind product targets of P_{80} 55, 75, 100, 125 and 180 µm. There were improvements to the recovery of gold with each successively finer grind until 75 µm was achieved, after which the recovery reached a plateau. This identified a grind P_{80} of 75 µm to be the preferred target. In the final flowsheet, ore was ground to a P_{80} of 75 µm with pyrite/arsenopyrite rougher flotation conducted using 200 g/t copper sulfate and 125 g/t PAX. MIBC was used as the frother.

These flowsheets were then used to test the response to antimony rougher flotation of ten Yellow Pine high antimony variability samples, and gold rougher flotation on a suite of nineteen variability samples. Results from those tests are shown below in Table 13.5 and Table 13.6.

Table 13.5: Yellow Pine Antimony Rougher Flotation Recoveries

Composite/ Test ID	Feed Grade (calc)				Concentrate Grade				Recovery			
	Au (g/t)	As (%)	Sb (%)	S (%)	Au (g/t)	As (%)	Sb (%)	S (%)	Au (%)	As (%)	Sb (%)	S (%)
YPH-CF22	1.98	0.25	0.33	1.04	12.56	1.38	10.71	9.63	17.0	14.6	87.3	24.9
YPH-BF13	2.11	0.25	0.39	1.06	7.56	0.71	10.50	6.92	12.0	9.4	89.3	21.8
YP105	1.66	0.23	0.76	1.63	2.66	0.26	9.31	6.15	9.5	6.5	73.1	22.4
YP107	1.21	0.39	0.11	1.24	1.52	0.49	2.57	2.42	3.0	3.0	55.3	4.7
HSTK139	2.27	0.33	0.29	1.64	3.10	0.34	4.52	3.15	7.4	5.7	84.8	10.4
HSTK144	1.82	0.28	0.08	1.05	5.77	0.97	2.27	3.22	8.9	9.6	76.5	8.6
EP5	2.66	0.55	0.44	1.31	9.75	1.75	10.44	7.51	13.8	12.1	89.5	21.6
EP6	3.29	0.55	0.17	1.67	16.52	2.95	7.85	10.10	8.3	8.9	76.7	10.0
EP7	3.52	0.31	0.97	1.69	10.64	0.64	19.13	10.26	14.7	9.9	96.0	29.5
EPH	2.51	0.46	0.68	1.58	5.38	0.89	13.24	7.53	10.2	9.4	92.6	22.7
Average	2.30	0.36	0.42	1.39	7.55	1.04	9.05	6.69	10.5	8.9	82.1	17.7

Rougher recoveries include the gold recovered to the antimony rougher concentrate, much of which is ultimately diverted to the gold circuit feed through the antimony cleaner tailings.

Table 13.6: Yellow Pine Gold Rougher Flotation Recoveries

Composite/ Test ID	Head Grade (calc)			Concentrate Grade			Unit Recovery		
	Au (g/t)	As (%)	S (%)	Au (g/t)	As (%)	S (%)	Au (%)	As (%)	S (%)
EP1	2.67	0.55	1.25	15.13	3.09	7.40	89.6	88.7	93.9
EP2	0.97	0.47	1.20	5.47	2.82	7.28	90.5	95.9	97.2
EP3	2.71	0.21	0.92	20.59	1.55	7.02	94.5	94.0	95.2
EP4	2.31	0.25	0.89	17.85	1.90	6.84	96.2	95.1	96.1
EP5	2.66	0.55	1.31	11.74	2.42	5.39	93.2	91.7	95.4
EP6(1)	3.29	0.55	1.67	14.35	2.29	7.20	94.5	91.8	95.3
EP7(1)	3.52	0.31	1.69	14.09	1.28	5.67	93.5	90.4	95.0
EPH-CL1(1)	2.51	0.46	1.58	11.64	2.15	6.54	92.1	92.9	95.6
EPH-LCT(1)	3.24	0.45	1.59	16.01	2.26	6.92	95.0	93.6	96.4
EPL	1.73	0.40	1.13	8.65	1.98	5.80	92.0	92.4	94.9
HSTK139(1)	2.27	0.33	1.64	9.03	1.31	6.79	89.1	88.5	96.0
HSTK144(1)	1.82	0.28	1.05	7.67	1.17	4.66	90.7	89.1	95.2
YP107(1)	1.21	0.39	1.24	4.05	1.34	4.44	79.6	80.3	86.7
YP108	1.89	0.48	1.25	11.35	2.78	7.56	93.8	91.1	94.6
YP119	2.72	0.45	1.55	10.13	1.62	5.95	91.1	87.3	93.7
YP130	2.28	0.38	0.86	16.40	2.68	6.30	95.8	94.7	98.0
YPH-CF22(1)	1.98	0.25	1.04	9.47	1.22	4.46	92.1	89.5	91.6
YPH-LCT1(1)	2.11	0.25	1.06	11.00	1.29	4.85	94.4	90.3	95.4
YPL	1.86	0.36	1.01	14.03	2.69	7.65	94.3	92.5	94.8
Averages	2.30	0.39	1.26	12.03	1.99	6.25	92.2	91.1	94.8

Note:
(1) High Sb samples.

In this study, only limited locked cycle cleaner testing has been performed as (a) antimony is not the primary metal of interest in the study and (b) gold flotation for the most part would probably not include closed circuit cleaning. However, to demonstrate the recovery of antimony in closed circuit cleaning, and to explore the potential for higher recovery of gold to the gold concentrates, the key composites have been tested in locked cycle mode. In the case of the YPH and EPH composites, for locked cycle testing, in addition to antimony cleaning as described earlier, the gold rougher concentrate was then open-circuit cleaned in a single stage with 10 g/t PAX followed by scavenging with a further 10 g/t PAX. The pyrite/arsenopyrite cleaner scavenger tailings were directed to final tailings. The results from a six cycle locked cycle test on each of the YPH and EPH composites are presented in Table 13.7, along with the results of one batch YPH test which produced concentrate meeting the revised pressure oxidation (POX) feed target.

Antimony cleaner recoveries were 86% and 91% to concentrates assaying 57% and 62% antimony and 9 and 11 g/t gold. The gold lost to the antimony product was 2.5% and 3.1%. The gold cleaner concentrates assayed 25.0 and 34.4 g/t Au, and 11.1% and 15% S, respectively. Overall gold recoveries to the cleaner concentrates were 88.3% and 88.7%. The gold rougher concentrates assayed from 11 g/t to 16 g/t Au, and from 4.9% to 6.9% S. Overall gold recoveries to the locked cycle test rougher concentrates were 92% for both tests, the batch test recovery was quite low but expected to increase significantly with the closed cycle operation of the antimony circuit.

Table 13.7: Antimony and Gold Flotation from Yellow Pine High Antimony Samples

High Sb Samples	Weight		Assays					Distribution			
	Dry	%	Au (g/t)	As (%)	Sb (%)	S (%)	CO_3 (%)	Au (%)	As (%)	Sb (%)	S (%)
Yellow Pine High Sb											
LCT Sb Final Concentrate	11.4	0.57	9.28	0.19	57.23	28.10	n/a	2.5	0.4	86.3	15.1
LCT Au Cleaner Concentrate	149.0	7.4	25.03	2.84	0.48	11.08	3.55	88.3	83.8	9.5	78.1
LCT Au Rougher Concentrate	353.6	17.7	11.00	1.29	0.24	4.85	2.59	92.1	90.0	11.5	81.0
LCT Au Rougher Tailings	1638	81.8	0.14	0.03	0.01	0.05	0.80	5.4	9.6	2.2	3.9
LCT Au Rougher + Cleaner Tailings	1843	92.0	0.21	0.04	0.02	0.08	0.92	9.2	15.7	4.2	6.8
BT Sb Rougher Concentrate	76.1	3.8	8.19	0.50	8.73	6.50	n/a	14.7	7.4	81.8	23.3
BT Au Rougher Concentrate	302	15.1	11.06	1.42	0.38	5.02	1.25	78.5	83.2	14.2	71.3
BT Au Rougher Tailings	1625	81.1	0.18	0.03	0.02	0.07	n/a	6.9	9.4	4.0	5.4
Early Production High Sb											
LCT Sb Final Concentrate	19.1	0.95	10.50	0.33	62.10	26.60	n/a	3.1	0.7	91.1	15.9
LCT Au Cleaner Concentrate	168.1	8.3	34.40	4.78	0.59	15.00	2.95	88.7	87.9	7.6	78.7
LCT Au Rougher Concentrate	374.9	18.6	16.00	2.26	0.29	6.92	n/a	92.0	92.9	8.3	81.0
LCT Au Rougher Tailings	1619	80.4	0.20	0.04	0.01	0.06	0.85	4.9	6.4	0.6	3.0

Note: LCT - Locked Cycle Test, BT - Batch Test

In locked cycle testing of the YPL and EPL composites, the rougher concentrates were cleaned in a single stage with 10 g/t PAX added to the last one-third of the bank. Cleaner tailings were reground with 10 g/t copper sulfate and floated with 10 g/t PAX to scavenge an additional 2% of gold from the cleaner tailings. The gold flotation results from six cycle locked cycle tests on the YPL and EPL composites are presented in Table 13.8. The final cleaner concentrate gold grades were 35.4 g/t and 28.7 g/t and the sulfur grades were 19.1% and 14.6%. In both cases, the overall gold recovery to the cleaner concentrate was just over 92%. The rougher concentrate gold grade was 10.7 g/t and the sulfur grade was 5.7%, 93.3% of the gold reported to the rougher concentrate.

Table 13.8: Gold Flotation from Low-Antimony Yellow Pine Samples

Low Sb Samples	Weight		Assays				Distribution		
	Dry	%	Au (g/t)	As (%)	S (%)	CO_3 (%)	Au (%)	As (%)	S (%)
Yellow Pine Low Sb									
LCT Au Cleaner + Scavenger Concentrate	89.9	4.8	35.4	6.89	19.1	1.20	92.3	89.8	93.3
LCT Au Rougher + Scavenger Tailings	1766	95.2	0.15	0.04	0.07	n/a	7.7	10.2	6.7
BT Au Rougher Concentrate	577.6	14.5	10.7	2.07	5.7	1.85	93.3	92.8	94.2
BT Au Rougher Tailings	3414	85.5	0.13	0.03	0.06	n/a	6.7	7.2	5.8
Early Production Low Sb									
Au Cleaner + Scavenger Concentrate	133.1	7.1	28.7	4.99	14.6	1.20	92.2	88.5	94.4
Au Rougher + Scavenger Tailings	1730	92.9	0.19	0.05	0.07	1.50	7.8	11.5	5.6

Note: LCT - Locked Cycle Test, BT - Batch Test

The final antimony concentrate and gold rougher and cleaner concentrates from the Yellow Pine Low and Yellow Pine High tests were subjected to full Inductively Coupled Plasma, mercury, halides and whole rock analyses, the results from which are shown in Table 13.9.

Table 13.9: ICP, WRA, and Halide Analysis of Yellow Pine Concentrates

Analyte	Units	YPH Sb Cleaner Con	YPH Au Cleaner Con	YPH Au Rougher Con	YPL Au Cleaner Con	YPL Au Rougher Con	Analyte	Units	YPH Sb Cleaner Con	YPH Au Cleaner Con	YPH Au Rougher Con	YPL Au Cleaner Con	YPL Au Rougher Con
Al	%	0.55	5.54	7.43	5.04	8.16	CO_3	%		3.55	1.25	1.20	1.85
As	ppm	4120	31000	13700	66000	21000							
Ba	ppm	50	510	660	330	610	LOI	%	49.60	9.82	4.67	18.70	7.27
Be	ppm	<5	6	<5	10	7	Al_2O_3	%	1.16	9.78	14.30	9.85	16.00
Ca	%	0.2	1.7	0.7	0.6	1.1	CaO	%	0.3	2.2	0.9	0.9	1.4
Cd	ppm	20	<10	0	<10	1	Cr_2O_3	%	0.010	1.020	0.440	0.870	0.420
Cr	ppm	60	7510	3000	110	2800	Fe_2O_3	%	4.070	19.000	8.730	30.800	10.500
Co	ppm	<10	80	36	5810	37	K_2O	%	0.8	6.0	8.1	4.8	8.1
Cu	ppm	280	970	680	1050	480	MgO	%	0.12	0.98	0.98	0.76	1.09
Fe	%	2.64	14.30	6.01	22.00	7.29	MnO	%	0.014	0.144	0.060	0.100	0.070
K	%	0.6	5.3	6.7	4.0	6.6	Na_2O	%	n/a	n/a	0.190	0.120	0.170
La	ppm	10	100	59	100	90	P_2O_5	%	0.06	0.22	0.12	0.13	0.15
Li	ppm	20	<10	10	<10	10	SiO_2	%	5.87	42.70	58.00	30.50	52.50
Mg	%	0.08	0.64	0.44	0.37	0.54	TiO_2	%	0.06	0.99	0.51	1.50	0.75
Mn	ppm	100	1190	490	660	500	V_2O_5	%	0.002	0.019	<0.01	0.020	0.020
Mo	ppm	30	130	67	90	50	SUM	%	62.0	92.9	97.0	99.1	98.4
Ni	ppm	<10	3560	1450	2560	1320							
P	%	0.02	0.10	0.06	0.05	0.06	Hg	ppm	252.0	5.23	3.01	11.90	3.72
Pb	ppm	2290	680	599	30	76	Se	ppm	n/a	n/a	3	n/a	2
Sb	ppm	581952	4540	3140	3600	951	Bi	ppm	n/a	n/a	0.2	n/a	0.2
Sc	ppm	<5	6	<5	<5	<5	Ag	ppm	353.0	n/a	9.7	n/a	9.8
Sn	ppm	<50	<50	7	<50	9							
Sr	ppm	20	230	140	130	170	F	%	n/a	n/a	0.1450	n/a	0.1310
Ti	%	0.04	0.64	0.31	0.83	0.46	Cl	ppm	n/a	n/a	150	n/a	150
V	ppm	<10	110	60	110	64							
W	ppm	100	1840	408	230	112							
Y	ppm	<5	37	21	20	27							
Zn	ppm	2600	190	85	390	266							

13.5.3 Gold Concentrate Upgrading for Sales

Limited batch upgrading testing of gold concentrates from the two Yellow Pine composites for third-party sales indicated that it is possible to clean from rougher concentrates grading 11 g/t Au to concentrates grading 48 - 49 g/t Au in three stages of cleaning. Cleaning losses from the YPH tests were only 4.1%, however 14.7% of the gold was tied up in the antimony rougher concentrate, much of which would be rejected back to the gold circuit feed in locked cycle operation and its behavior in gold cleaning is unknown, while 15% in the gold was lost in cleaning the YPL rougher concentrate to a grade of 48 g/t.

13.5.4 Leaching of Flotation Tailings

Cyanidation leaches were conducted on the rougher or combined rougher/cleaner flotation tailings from tests on thirteen variability composites from Yellow Pine. A cyanide concentration of 1.25 g/L was used and for most tests the leach time was 48 hours; however for tests where kinetics samples were taken, the leach was complete within 10 hours (Gajo, 2014b). The results are summarized in Table 13.10.

Table 13.10: Yellow Pine Flotation Tailings Leach Extractions

Composite ID	Leach Feed Type	Gold Grade		Gold Recovery (%)
		Feed (g/t)	Residue (g/t)	
EP1	Rougher Tailings	0.28	0.25	11.8
EP2	Rougher Tailings	0.13	0.12	9.7
EP3	Rougher Tailings	0.20	0.18	10.7
EP4	Rougher Tailings	0.13	0.12	8.6
EP5*	Rougher Tailings	0.19	0.17	11.6
EP6*	Rougher Tailings	0.18	0.17	7.6
EP7*	Rougher Tailings	0.25	0.23	6.5
YP107* (cleaner)	Rougher + Cleaner Tailings	0.38	0.36	4.2
YP108	Rougher Tailings	0.13	0.11	16.9
YP119	Rougher Tailings	0.33	0.30	10.0
YP130	Rougher Tailings	0.11	0.09	14.3
HSTK144 (cleaner)(1)	Rougher + Cleaner Tailings	0.26	0.24	6.3
HSTK139*	Rougher Tailings	0.20	0.18	11.9
	Averages	0.21	0.19	10.0

Note:
(1) High Sb Samples.

13.5.5 Pressure Oxidation of Concentrates and Cyanide Leaching of Residues

During the PEA phase, three pressure oxidation tests were run at Dynatec Metallurgical Technologies (Dynatec) on Yellow Pine gold flotation concentrate produced during that program. Neither regrinding nor acidulation of the concentrate was employed prior to oxidation testing. The concentrate, grading 28.9 g/t Au, 12% S and 1.6% CO_3, was tested at 200, 215 and 230 °C each with samples taken at 40, 60 and 80 minutes. Sulfur oxidation was very rapid with oxidation essentially complete within 40 minutes for the two highest temperatures tested (Masters, 2012).

The residues were then leached at McClelland Laboratories through bottle roll leaching “as is” and after regrinding to P_{80} 45 µm. The results indicated that the gold was readily leached into solution, with recoveries of 96% for the “as is” leach and 99% for the reground residues. The silver did not leach well. Cyanide consumption in the leaches ranged from 3.9 kilograms per metric tonne (kg/t) for the “as is” leach and 4.3 kg/t for the reground residue leach (McClelland, 2012).

In the PFS phase of testwork, confirmation pressure oxidation tests were conducted at SGS Lakefield on bulk cleaner concentrate produced from a 13 cycle locked cycle test on low antimony Yellow Pine Composite P2 (Jackman, 2014a). One oxidation test was also conducted on remaining Sub-A3 concentrate from the PEA study (Jackman, 2014b). In total, four pressure oxidation tests were conducted at 220°C under 75 psi oxygen (O_2) for 60 minutes, with two tests acidulating the concentrates prior to POX and the final two using no acidulation to better mimic plant conditions as recommended by Dynatec. Sulfide oxidation averaged 97.5% in the acidulated tests and averaged 98.7% in the non-acidulated tests. The resulting slurries were then conditioned in a hot cure tank at 95 °C for 2 hours prior to being forwarded on for further testing. Hot cure solution assays showed similar average arsenic in the test samples at 2.9 ppm acidulated and 2.8 ppm non-acidulated. These results confirmed the recommended autoclave conditions as presented by Dynatec in the PEA program and that acidulation was not needed for the Yellow Pine concentrates.

Cyanide leaching of the PFS program POX slurries was completed at SGS Lakefield and evaluated baseline cyanidation extraction and effect of use of flotation tailings in slurry neutralization (Jackman, 2014a; Jackman, 2014b). Residue samples from three of the PFS program pressure oxidation runs underwent six cyanidation leaches, without regrinding, to determine the extraction of gold. For the three parallel straight cyanidation leaches after washing, the average gold extraction was 98.9% and silver extraction was negligible, while consumptions averaged 0.57 kg NaCN per tonne of POX solids and 21.7 kg lime per tonne of POX solids. There appears to have been anomalously high lime consumption for the Sub-A3 POX cyanidation, so it is possible that the actual lime consumption would be lower. It was noted that the recovery of silver may be enhanced by using different treatment methods on the POX discharge such as a lime boil rather than hot cure.

The effect of using flotation tailings versus limestone for stage 1 neutralization (to pH 4.5) prior to cyanidation on partially washed POX discharge slurry (34% PLS and 66% distilled, deionized water) was also evaluated. It was found that essentially all of the equivalent mass of tailings from the flotation test would be required to bring the washed POX slurry to a pH of 4.5 in stage one of neutralization, as compared to 174 g of limestone per liter. The subsequent amount of lime needed to bring both slurries from pH 4.5 up to pH 10.0 was similar at 31.6 and 25.7 g/L. Both neutralization approaches yielded 97.8% gold extraction in subsequent cyanidation.

13.5.6 Yellow Pine Oxides

Leaching of Yellow Pine oxides was not tested in the PFS program, but has been studied in the past (in addition to the heap leach operation previously described); using a grind of P_{100} 150 µm, those tests yielded gold extractions of 84% to 89% (Albert, 1997). It is generally believed that almost all Yellow Pine oxides have already been processed so existing oxide mineralized material is minimal.

13.6 HANGAR FLATS DEPOSIT

13.6.1 Historical Metallurgy

Hangar Flats ores were milled through the Meadow Creek mill from 1932 to 1938. Only high-grade antimony ore was floated during that period, assaying over 4% antimony in the mill feed.

Of all the testwork completed on ores from the area in the past few decades, no testwork has been identified that dealt specifically with the Hangar Flats/Meadow Creek area ores.

13.6.2 Flotation

The Hangar Flats High antimony (HFH) composite was blended to represent the average feed grade of gold, antimony and sulfur expected over the four phases of mining as set forth in the PEA.

Grind optimization testwork was completed on the other, low antimony composites, and applied to the HFH composite. As mentioned in the Yellow Pine antimony-gold bearing section, early testwork on the new high antimony composites (both Hangar Flats and Yellow Pine) indicated the need for stainless steel grinding to maintain higher pulp potentials for good stibnite flotation (Gajo, 2014a).

In the final flowsheet on Hangar Flats antimony-rich material, ore was ground to a product size of P_{80} 75 µm with 200 g/t lime and 75 g/t sodium cyanide, stibnite was floated using 355 g/t lead nitrate and 15 g/t Aerophine 3418A, then cleaned without regrinding using 10 g/t sodium cyanide in the first of two stages of cleaning and 5 g/t Aerophine 3418A in the final stage of cleaning. The subsequent pyrite/arsenopyrite rougher flotation was conducted using 275 g/t copper sulfate and 150 g/t PAX then cleaned in a single stage with 10 g/t PAX followed by scavenging with a further 5 g/t PAX. MIBC was used as the frother throughout. Results of the batch antimony rougher flotation tests conducted with this flowsheet on six Hangar Flats antimony-bearing samples are shown in Table 13.11 below. The average antimony rougher flotation recovery was 84%.

Table 13.11: Hangar Flats Antimony Rougher Flotation Results

Composite/ Test ID	Feed Grade (calc)				Concentrate Grade				Recovery			
	Au (g/t)	As (%)	Sb (%)	S (%)	Au (g/t)	As (%)	Sb (%)	S (%)	Au (%)	As (%)	Sb (%)	S (%)
HFH-BF17	1.86	0.39	0.48	1.06	2.80	0.47	8.55	4.35	7.0	5.5	83.1	18.9
HFH-CF22	1.83	0.36	0.45	1.07	2.91	0.47	9.85	5.40	5.7	4.6	79.4	18.1
HF33	1.20	0.42	0.14	2.19	1.73	0.59	4.42	3.87	3.3	3.2	75.0	4.0
HF38	1.43	0.39	0.17	0.82	4.04	1.04	4.39	3.59	8.5	7.1	77.3	11.8
HF43	2.61	0.24	0.88	1.63	3.61	0.27	12.74	6.41	8.5	7.2	89.3	24.2
Comp A(1)	2.56	0.50	0.40	1.29	7.15	1.23	6.19	5.07	16.4	14.6	90.6	23.2
Average	1.91	0.38	0.42	1.35	5.10	1.07	7.87	5.51	10.9	9.8	83.9	19.0

Note:
(1) Contained 52% Hangar Flats material.

The Hangar Flats Low antimony (HFL) composite for PFS testwork was blended to represent the average feed grades of gold and sulfur expected over the four phases of mining as set forth in the PEA. This composite had a target feed sulfur grade of 0.8% which brings it just bordering the transitional ore range. It was expected to have slightly lower gold flotation recoveries and elevated tailings leach gold extractions than higher sulfide containing ores.

Applying the PEA flowsheet to the HFL composite resulted in the production of a rougher concentrate grading 12.8 g/t Au, 7.4% S and recovered 81.4% of the gold into that rougher concentrate. A single stage of cleaning upgraded the concentrate to 28.3 g/t Au and 16.9% S with a cleaner stage recovery of 94.3% of the gold for an overall gold recovery of 76.8%.

Grind optimization testwork was completed by testing the flotation response of gold at primary grind targets P_{80} of 55, 75, 100, 125 and 150 µm. There did not to appear to be any improvement to the recovery of gold with each successively finer grind so the previously identified grind P_{80} of 75 µm preferred for Yellow Pine and West End (these two deposits containing the majority of recoverable gold in the Project) was chosen as the target for the Hangar Flats program.

Reagent optimization testwork was then undertaken to identify the maximum amount of gold to be recovered from the HFL composite with the optimum dosages of reagents. It was found with the new composite that the sodium silicate use could be discontinued, however copper sulfate dosage required doubling from 140 g/t to 250 g/t and xanthate was increased by 35% from 140 g/t to 200 g/t to maximize the rougher recovery of gold. The rougher concentrate was cleaned in a single stage with 60 g/t PAX, added in one-third increments through the cleaner. Cleaner tailings were reground with 10g/t copper sulfate and floated with 10 g/t PAX to recover an additional 2% of gold from the cleaner tailings. MIBC was used as the frother throughout. It is recommended that for the plant environment, a cyclone be used to return the coarse fraction of the cleaner tailings to the primary ball mill for reprocessing with the rougher stream. Batch rougher testwork on the HFL composite using this flowsheet produced a rougher concentrate which averaged 9.8 g/t Au and 5.4% S at a mass pull of 14% and recovery of 85.8% of the gold.

Table 13.12: Hangar Flats Gold Rougher Flotation Results

Composite/ Test ID	Head Grade (calc)			Concentrate Grade			Unit Recovery		
	Au (g/t)	As (%)	S (%)	Au (g/t)	As (%)	S (%)	Au (%)	As (%)	S (%)
HFH-LCT1(1)	1.85	0.39	1.06	9.34	1.67	4.75	92.1	77.3	95.5
HFH-CF22	1.83	0.36	1.07	11.48	1.88	6.00	91.4	74.7	94.3
HF33(1)	1.20	0.42	2.19	4.80	1.87	11.35	73.7	82.2	95.8
HF38(1)	1.32	0.40	0.84	6.69	2.08	4.21	91.3	92.0	93.4
HF43(1)	2.99	0.23	1.61	13.08	0.86	5.78	90.1	77.2	89.5
HFL-CF10	1.58	0.44	0.80	8.73	2.04	4.78	86.1	72.3	93.6
HF28.1	1.14	0.55	1.13	5.37	2.62	5.46	92.2	92.9	94.3
HF31	1.46	0.59	1.42	4.15	1.62	4.46	74.2	72.4	82.3
HF42.1	0.97	0.39	0.76	6.17	2.56	5.00	91.2	93.9	94.4
Average (Sulfide)	1.59	0.42	1.21	7.76	1.91	5.75	86.9	81.6	92.6
HF32(2)	1.38	0.78	0.05	5.10	1.30	0.20	33.4	14.7	31.3
HF29.3(2)	1.18	0.39	0.16	4.20	0.90	0.60	37.1	23.5	39.8
HF29.2(2)	1.55	0.85	0.25	6.00	1.20	0.70	41.7	14.5	31.6
Comp F(2)	1.10	0.56	0.60	5.90	3.00	5.60	49.0	48.0	84.0
HF30.2(2)	1.64	0.70	0.74	9.10	2.50	5.40	61.0	39.5	79.7
Average (Oxide-Transition)	1.37	0.66	0.36	6.06	1.78	2.50	44.4	28.0	53.3

Notes: *
(1) High Sb samples
(2) Oxide and transitional material samples

Gold rougher flotation results from tests on nine sulfide samples, using the PFS flowsheet are shown in Table 13.12. The average gold recovery from the Hangar Flats sulfide samples was 87%. Data from five oxide-transition samples are also shown. These yielded inferior flotation recoveries, but gold invariably leached well from the tailings of these samples, averaging 80% extraction of remaining gold (Ratnayake, 2013c; Gajo, 2014c; Gajo, 2014b).

The results from a six cycle locked cycle test on the antimony-rich HFH composite are presented in the following table (Table 13.13). Antimony cleaner recovery was 80% to a concentrate assaying 58% antimony and 4.9 g/t gold, while the gold lost to the antimony product was 1.7%. This compared with 83% in the antimony batch rougher test suggests a cleaner stage recovery of 97%. The gold cleaner concentrate assayed 28.2 g/t gold and 14.5% S and had an overall gold recovery of 86.3% with a cleaner stage recovery of gold of 95.3%. The gold rougher concentrates from the locked cycle and batch tests assayed from 9.3 to 9.5 g/t Au and 4.8% to 5.3% S. Overall gold recovery to the locked cycle test gold rougher concentrate was 90.6%.

Some 14% of the antimony reported to the gold concentrate indicating possible upside in this recovery to the antimony concentrate with further refining of the antimony rougher flotation circuit.

Table 13.13: Antimony and Gold Flotation from the Hangar Flats High Antimony Composite

Hangar Flats High Sb	Weight		Assays					Distribution			
	Dry	%	Au (g/t)	As (%)	Sb (%)	S (%)	CO_3 (%)	Au (%)	As (%)	Sb (%)	S (%)
LCT Sb Final Concentrate	12.5	0.62	4.93	0.15	58.06	25.83	n/a	1.7	0.2	80.4	15.3
LCT Au Cleaner Concentrate	113.4	5.7	28.24	4.69	1.10	14.52	2.40	86.3	68.2	13.8	77.9
LCT Au Rougher Concentrate	359.7	18.0	9.34	1.67	0.40	4.75	2.06	90.6	77.1	16.0	80.9
LCT Au Rougher Tail	1629	81.4	0.18	0.11	0.02	0.05	1.35	7.8	22.7	3.6	3.9
LCT Au Rougher + Cleaner Tail	1875	93.7	0.24	0.13	0.03	0.08	1.42	12.0	31.6	5.8	6.8
BT Sb Rougher Concentrate	58.3	2.9	3.42	0.46	13.20	6.45	n/a	5.9	3.5	83.4	18.0
BT Au Rougher Concentrate	304	15.3	9.49	1.80	0.40	5.31	2.20	85.8	73.0	13.1	77.3
BT Au Rougher Tail	1630	81.8	0.17	0.11	0.02	0.06	n/a	8.2	23.5	3.5	4.7
Note: LCT - Locked Cycle Test, BT - Batch Test											

In locked cycle testing the HFL composite, the rougher concentrates were cleaned in a single stage with 50 g/t PAX added in two doses of 25g/t each down the bank, followed by scavenger flotation with an additional 10 g/t PAX. MIBC was used as frother throughout. Cleaner tailings were returned to the ball mill for reprocessing with fresh feed, which improved the rougher gold recovery over batch testing.

The gold flotation results from a six cycle locked cycle test on the HFL composite as well as a batch rougher without closed circuit cleaning are presented in Table 13.14. The final cleaner concentrate gold grade was 28.3 g/t and the sulfur grade was 14.9% with overall gold recovery to the cleaner concentrate just over 82%. The rougher concentrate gold grade was 10.5 g/t and the sulfur grade was 5.9% with 79.3% of the gold reporting to the rougher concentrate.

Table 13.14: Gold Flotation from Hangar Flats Low Antimony Composite

Hangar Flats Low Sb	Weight		Assays				Distribution		
	Dry	%	Au (g/t)	As (%)	S (%)	CO_3 (%)	Au (%)	As (%)	S (%)
LCT Au Cleaner + Scav Conc	88.2	4.8	28.3	6.13	14.91	1.15	82.9	63.3	89.7
LCT Au Rougher + Scav Tail	1740	95.2	0.30	0.18	0.087	n/a	17.1	36.7	10.3
BT Au Rougher Concentrate	253.6	12.7	10.5	2.60	5.9	1.20	79.3	64.3	92.4
BT Au Rougher Tail	1745	87.3	0.40	0.21	0.07	n/a	20.7	35.7	7.6
Note: LCT - Locked Cycle Test, BT - Batch Test									

Antimony cleaner concentrates and gold rougher and cleaner concentrates from the tests on HFH and HFL samples were subjected to full ICP, mercury, halide and whole rock analysis scans, the results from which are shown in Table 13.15.

Table 13.15: ICP, WRA and Halide Analysis of Hangar Flats Concentrates

Analyte	Units	HFH Sb Cleaner Con	HFH Au Cleaner Con	HFH Au Rougher Con	HFL Au Cleaner Con	HFL Au Rougher Con	Analyte	Units	HFH Sb Cleaner Con	HFH Au Cleaner Con	HFH Au Rougher Con	HFL Au Cleaner Con	HFL Au Rougher Con
Al	%	0.56	6.24	7.52	6.07	0.45	CO_3	%		2.40	2.20	1.15	1.20
As	ppm	1420	48600	18900	57800	>10000							
Ba	ppm	70	390	650	330	88	LOI	%	54.50	15.70	6.70	17.00	7.71
Be	ppm	<5	7	6	8	<5	Al_2O_3	%	1.09	12.50	14.40	11.50	16.40
Ca	%	0.6	1.0	1.2	0.6	0.6	CaO	%	0.80	1.50	1.52	0.90	1.13
Cd	ppm	<10	<10	0	<10	<10	Cr_2O_3	%	0.01	0.37	0.29	0.00	0.31
Cr	ppm	50	2400	2030	4690	2020	Fe_2O_3	%	1.42	23.70	9.79	26.50	11.20
Co	ppm	<10	50	31	70	34	K_2O	%	0.80	5.40	6.32	4.50	6.28
Cu	ppm	460	640	440	930	442	MgO	%	0.47	1.17	1.25	0.74	1.15
Fe	%	0.97	15.60	6.80	18.50	7.41	MnO	%	0.04	0.08	0.07	0.00	0.06
K	%	0.7	4.2	5.3	3.7	0.3	Na_2O	%	n/a	n/a	0.60	n/a	0.85
La	ppm	<10	100	89	100	40	P_2O_5	%	0.05	0.17	0.17	0.11	0.17
Li	ppm	<10	20	20	10	<10	SiO_2	%	6.59	42.00	54.90	35.40	51.40
Mg	%	0.30	0.67	0.64	0.45	0.23	TiO_2	%	0.05	1.39	0.79	1.54	0.97
Mn	ppm	290	610	520	640	393	V_2O_5	%	0.00	0.02	0.01	0.02	0.01
Mo	ppm	<10	40	37	70	48	SUM	%	65.8	104.0	96.8	98.2	97.6
Ni	ppm	<10	1080	969	2200	943							
P	%	0.02	0.07	0.06	0.05	0.07	Hg	ppm	342.00	33.10	15.30	67.60	38.00
Pb	ppm	1140	580	545	<20	22	Se	ppm	n/a	n/a	3.00	n/a	2
Sb	ppm	579566	11000	3280	5260	1830	Bi	ppm	n/a	n/a	0.2	n/a	0.2
Sc	ppm	<5	6	<5	6	<5	Ag	ppm	684.0	n/a	13.00	n/a	12.30
Sn	ppm	<50	<50	6	<50	<50							
Sr	ppm	40	220	250	200	81	F	%	n/a	n/a	0.1390	n/a	0.2280
Ti	%	0.03	0.79	0.47	0.92	<0.01	Cl	ppm	n/a	n/a	150.0	n/a	<50
V	ppm	<10	110	72	110	23							
W	ppm	<50	<50	66	250	<50							
Y	ppm	<5	17	18	21	10							
Zn	ppm	1540	210	76	330	144							

13.6.3 Gold Concentrate Upgrading for Sales

Limited batch testing on the HFH and HFL composites of gold concentrate upgrading to explore the option of direct sales of gold concentrate yielded concentrates grading 48.7 g/t and 37.2 g/t Au, and 27.7% and 24.7% S respectively. Cleaner losses were 8.3% and 18.5% for the HFH and HFL composites. Further testing may improve this, although the mineralogy points to the finest disseminated pyrite and arsenopyrite (*i.e.* those preferentially rejected in cleaning) being the most enriched in gold, so cleaner performance to high-grade samples may always be relatively poor.

13.6.4 Leaching of Flotation Tailings

The tailings from bulk flotation tests conducted on HFH and HFL composites were subjected to scoping cyanide leaching tests (Gajo, 2014c). For the high antimony composite (HFH) cyanidation extracted 36% of the gold in the tailings, with the recovery effectively representing roughly 0.12 g/t recoverable gold. For the low antimony sample, which at 0.81% sulfur in feed is considered bordering transitional, 68% of the gold remaining in the tailings was extracted, representing a highly economic 0.19 g/t recoverable gold. Leaching of flotation tailings on six additional sulfide variability samples showed a range of recoveries from 7% to 19%, extracting from 0.01 to 0.06 g/t of gold.

Based on the scoping results, a developmental leach study was performed which utilized the combined rougher + cleaner tailings from the HFL locked cycle test. Results of a matrix of tests evaluating three different feed percent solids and three different cyanide concentrations identified the optimum gold extraction from the HFL tailings as being 53.4% of the gold contained in the tailings. A batch agitated tank carbon-in-pulp (**CIP**) leach was then conducted at 45% solids and 0.25 kg cyanide per tonne of tailings, which resulted in a gold extraction of 53.9%.

The combined gold recoveries from the locked cycle test and tailings leach for the HFH and HFL composites were calculated to be 91% and 92%, respectively.

13.6.5 Pressure Oxidation of Concentrates and Cyanide Leaching of Residues

During the PEA phase, three pressure oxidation tests were run at Dynatec in Fort Saskatchewan, Alberta on Hangar Flats gold flotation concentrates produced from the global Sub-A1 composite. Neither regrinding nor acidulation was used in these tests. The concentrate, grading 27.1 g/t Au, 11.8% S and 1.8% CO_3, was tested at 200, 215 and 230 °C each with samples taken at 40, 60 and 80 minutes. Sulfur oxidation was very rapid with oxidation essentially complete within 40 minutes for the two highest temperatures tested (Masters, 2012).

The residues were leached at McClelland Laboratories through bottle roll leaching "as is" and after regrinding to P_{80} 45µm. The total residence time in the leach was 72 hours. The results indicated that the gold was readily leached, with recoveries of 96% "as is" and 97% on reground residues. The silver extraction was less than 1% in both cases. Cyanide consumption in the tests was lower than with Yellow Pine, at 3 kg/t for the "as is" leach and 3.2 kg/t for the reground residue leach (McClelland, 2012).

In the PFS phase of testwork, a confirmation pressure oxidation test was completed on the remaining Sub-A1 concentrate that had been used for the Dynatec PEA study (Jackman, 2014b), using conditions recommended in Dynatec's report. The oxidation products were forwarded on for neutralization, CIP, cyanide destruction (CND), and environmental studies. Tests on the samples were conducted at 220 °C under 75 pounds per square inch (**psi**) of O_2 for 60 minutes, and included acidulation of the concentrates to pH1.8 prior to POX per internal SGS operating procedures. Sulfide oxidation averaged 95.5% in the tests, with POX solutions containing 2.57 ppm arsenic and 0.7 mg/L antimony. The results confirmed the Dynatec autoclave operating conditions, but future lab tests should use the full test recommendations from Dynatec and not acidulate the feed.

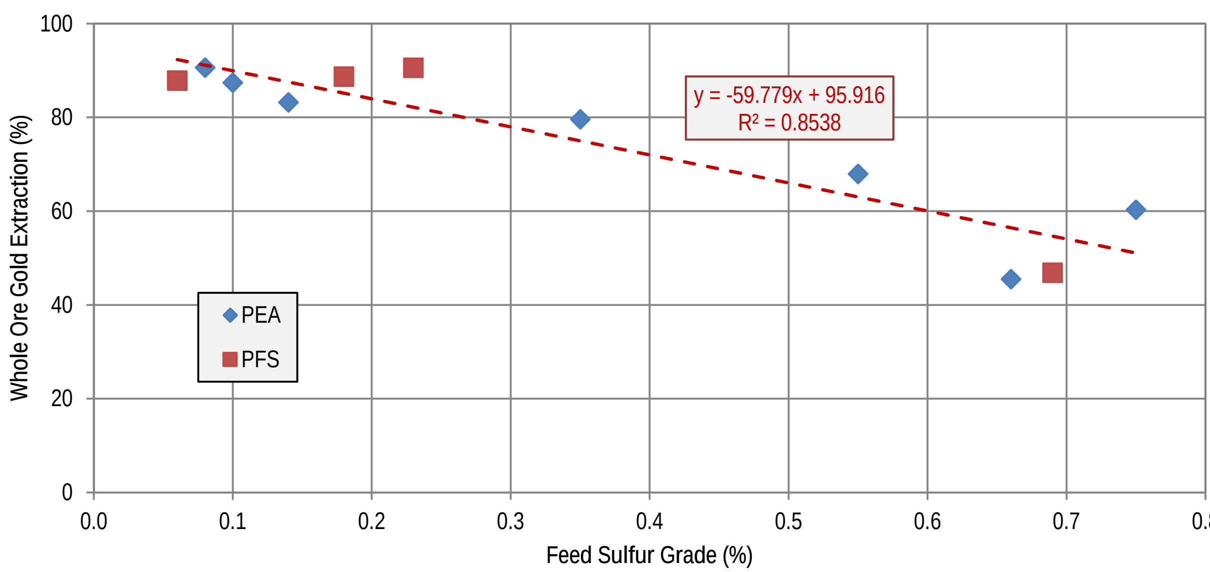
The samples from the three PFS program pressure oxidation runs were combined, fully washed and underwent a single CIP cyanidation leach to determine the extraction of gold (Jackman, 2014b). There were no regrinding tests performed. The gold extraction from the test was 97.8% and silver extraction was about 9%, while consumptions averaged 0.28 kg NaCN per tonne of POX solids and lime averaged 11.8 kg lime per tonne of POX solids. Again, cyanide consumption was lower with Hangar Flats POX residue than was seen with Yellow Pine.

13.6.6 Hangar Flats Oxides

There is a small amount of oxide mineralized material in the Hangar Flats Deposit and no past processing or testing reports have been found to indicate it has been worked or studied. Furthermore, no mineralogical gold balancing studies have been conducted on oxide samples from Hangar Flats, but it is believed that the gold is fine and mostly discrete in the true oxide materials.

During the PEA phase, whole ore cyanidation leaching was conducted on various oxide and transition samples with a standard diagnostic cyanidation procedure. Leach extraction was found to correlate quite closely to sulfide content. Cyanide consumption was 1.9 kg/t, while the lime consumption averaged 3.1 kg/t in these tests (Ratnayake, 2013c). An additional four oxide and transition samples were tested in the PFS phase to build upon that data set, and the combined results are shown on Figure 13.4. Consumption of cyanide for the new samples averaged 1.8 kg/t, while the lime consumption averaged 3.3 kg/t, quite similar to the PEA sample leaches (Gajo, 2014b).

Figure 13.4: Leach Gold Extraction *vs* Sulfur Grade for Hangar Flats Oxide and Transition Samples



13.7 WEST END DEPOSIT

13.7.1 Historical Metallurgy

Oxide ores from the West End Deposit were treated by heap leaching from 1982 to 1996. The heap leach process on ores (typically assaying roughly 1.3 g/t gold) involved crushing to minus 1.25 inches (32 mm) and a heap leach cycle of 50 days. Based on the information currently available, West End sulfides have never been processed commercially.

Testing conducted in the late 1970s and 1980s at Britton Research and Coastech Research on the sulfide ores yielded total recoveries in the 70% to 80% range through a combination of flotation, sulfide oxidation (roasting or bioleaching) and cyanidation of both the flotation tailings and oxidation residues (Britton, 1978; Broughton, 1987). Limited testing on oxide ores at Kappes, Cassiday and Associates occurred in 1997 which consisted of basic bottle roll extractions and all achieved between 84% and 89% extraction of gold (Albert, 1997).

13.7.2 Flotation

The flotation of West End sulfides is relatively straightforward, however very little flotation optimization work was conducted during the PEA phase. Most of the work constituted flotation from two global composites, namely A2 and Sub-A2, to create concentrate for downstream oxidation testing. The moderate levels of muscovite/illite in the composites facilitated the production of what was deemed a POX-ready flotation concentrate after roughing alone, recovering 83.6% and 81.8% of the feed gold to rougher concentrates assaying 9.4% and 9.9% sulfur respectively. This was achieved by a bulk sulfide float at a grind P_{80} target of 100 µm with 200 g/t copper sulfate and 110 g/t PAX. A single stage of cleaning of the A2 concentrate was able to reach grades of 30.7 g/t Au and 20.8% S, at a cleaner stage gold recovery of 93.2% for an overall gold recovery of 77.9%.

It was found in the PEA program oxidation study that the level of carbonates in the West End Sub-A2 concentrates necessitated the use of an acidulation step prior to autoclaving and this forced a change in the concentrate criteria for the PFS to include the carbonate/sulfur ratio (M3 has set a target of less than 0.9:1). Therefore, the primary recommendation from the PEA program was to investigate cleaning of the concentrates to reject carbonates in order to negate the need for the costly acidulation step.

The West End Sulfide (WES) composite for PFS testwork was blended to represent the average feed grade of gold expected over the four phases of mining as set forth in the PEA, but lacked a specific sulfur grade target. This composite's resulting sulfur grade of 0.65% brings it squarely into the transitional ore range. It was expected to have lower gold flotation recoveries and elevated tailings leach gold extractions than the higher sulfide containing ores. Applying the PEA flowsheet to the WES composite resulted in the production of a rougher concentrate grading 11.0 g/t Au, 5.1% S and 8.7% CO_3; and recovered 79.7% of the gold into that rougher concentrate at a mass pull of 12.2%. A single stage of cleaning upgraded the concentrate to 27.2 g/t Au, 13.2% S and 8.6% CO_3 with a cleaner stage recovery of 93.1% of the gold for an overall gold recovery of 74.2%. The carbonate to sulfur ratio was taken from 1.71 to 0.65 in one stage of cleaning.

Grind optimization testwork was completed at primary grind product targets of P_{80} 55, 75, 100, 150 and 180 µm. There was improvement to the recovery of gold with each successively finer grind. The recovery of gold in the 150 µm to 180 µm range was similar, but improved in the 75 µm -100 µm range, and again with the 55 µm grind. Due to concerns with the capital and operating costs involved in grinding to 55 µm, as well as the applicability of the 75 µm target to Yellow Pine, the grind product of P_{80} 75 µm was also chosen as the target for the WES program.

PFS reagent optimization testwork was then undertaken to identify the maximum amount of gold to be recovered while rejecting the most carbonates (Gajo, 2014a). Analysis of the PEA program data showed that the Yellow Pine and Hangar Flats concentrates had a carbonate to sulfur ratio of 0.13 and 0.15, respectively, while the West End concentrate was 1.14:1 for the carbonate to sulfur ratio. This was higher than the 0.9:1 limit imposed by M3 for the PFS. Through the batch flotation optimization tests, it was found that in a single stage of cleaning, the carbonate to sulfur ratio in the cleaner concentrate could be successfully reduced by two thirds as compared to the rougher concentrate ratio while maintaining a cleaner stage recovery of 95%.

In the final flowsheet, the feed was ground to a P_{80} of 75 µm with pyrite/arsenopyrite flotation conducted using 200 g/t copper sulfate and 175 g/t PAX, then cleaned in a single stage with 50 g/t copper sulfate and 50 g/t PAX. Cleaner tailings were reground with 25 g/t copper sulfate and floated with 25 g/t PAX to recover an additional 3% of gold from the cleaner tailings. It was recommended that for the plant environment, a cyclone be used to return the coarse fraction of the cleaner tailings to the ball mill for reprocessing with the rougher stream. Batch testwork on the

WES composite based on that flowsheet produced an average rougher concentrate of 9.6 g/t Au, 4.5% S and 9.3% CO_3 at a mass pull of 13.6% and recovery of 79.3% of the gold, nearly the same as the PEA flowsheet recovery. Single stage cleaning plus regrind and scavenging of the cleaner tailings produced an average concentrate grading 28.3 g/t Au, 14.0% S and 8.8% CO_3 at a cleaner stage recovery of 94.9% and an overall gold recovery of 75.2%, slightly exceeding the PEA flowsheet recovery by 1%. This drops the carbonate to sulfur ratio from 2.0 to 0.63 with a single stage of cleaning.

The results of the subsequent six-cycle locked cycle tests, which utilized the recycle of cleaner tailings back into primary grind for additional processing, are presented in Table 13.16 with a comparative batch test. The cleaner concentrate gold grade was 29.5 - 29.7 g/t, the sulfur grade was 14.4 - 15.0% and the carbonate was 7.9 - 8.2%, both having a comfortable carbonate to sulfur ratio of 0.55 or less in the concentrate. The batch rougher tests that this LCT was designed from achieved an average 79.3% rougher recovery, which gives this test a cleaner stage gold recovery of 97.8%. The locked cycle test tailings were forwarded for cyanidation testing, detailed later in this section. It is expected that a portion of the gold losses to tailings are associated with non-sulfide gangue minerals and would be available for leach recovery.

Table 13.16: Gold Flotation of Sulfide Concentrate from West End Sulfide Samples

West End Sulfide	Weight		Assays				Distribution			CO_3:S Ratio
	Dry	%	Au (g/t)	As (%)	S (%)	CO_3 (%)	Au (%)	As (%)	S (%)	
LCT Au Cleaner + Scav Conc	80.1	4.4	29.70	3.87	14.37	7.89	77.5	81.3	93.0	0.55
LCT Au Rougher + Scav Tail	1739	95.6	0.40	0.04	0.05	9.14	22.5	18.7	7.0	
BT Au Cleaner + Scav Conc	82.3	4.1	29.53	4.20	15.01	8.16	73.5	78.0	92.4	0.54
BT Au Rougher Concentrate	261.4	13.1	10.13	1.41	4.85	9.24	80.0	83.1	94.8	1.91
BT Au Rougher Tail	1739	86.9	0.38	0.04	0.04		20.0	16.9	5.2	
BT Au Rougher + Scav Tail	1918	95.9	0.46	0.05	0.05		26.5	22.0	7.6	
Note: LCT - Locked Cycle Test, BT - Batch Test										

Gold cleaner concentrates from the test were subjected to full ICP, mercury and whole rock analysis scans, the results from which are shown in Table 13.17.

Table 13.17: ICP, WRA and Halide Analysis of West End Sulfide Gold Cleaner Concentrate

Analyte	Units	WES Au Cleaner Con	Analyte	Units	WES Au Cleaner Con	Analyte	Units	WES Au Cleaner Con	Analyte	Units	WES Au Cleaner Con
Al	%	4.81	Mn	ppm	690	LOI	%	16.50	CO_3	%	7.69
As	ppm	37500	Mo	ppm	70	Al_2O_3	%	9.09	CO_3:S	none	0.55
Ba	ppm	310	Ni	ppm	2210	CaO	%	4.70			
Be	ppm	6	P	%	0.01	Cr_2O_3	%	<.001	Hg	ppm	19.00
Ca	%	3.3	Pb	ppm	20	Fe_2O_3	%	24.80	Se	ppm	n/a
Cd	ppm	<10	Sb	ppm	380	K_2O	%	4.50	Bi	ppm	n/a
Cr	ppm	4300	Sc	ppm	7	MgO	%	1.93	Ag	ppm	n/a
Co	ppm	170	Sn	ppm	<50	MnO	%	<.001			
Cu	ppm	990	Sr	ppm	130	Na_2O	%	n/a	F	%	n/a
Fe	%	17.40	Ti	%	0.26	P_2O_5	%	0.03	Cl	ppm	n/a
K	%	3.7	V	ppm	90	SiO_2	%	36.00			
La	ppm	40	W	ppm	100	TiO_2	%	0.44			
Li	ppm	30	Y	ppm	17	V_2O_5	%	0.02			
Mg	%	1.16	Zn	ppm	1430	SUM	%	98.0			

13.7.3 Gold Concentrate Upgrading for Sales

Limited testing using the WES composite on upgrading the concentrate for shipment to third party roasters and autoclaves has also been conducted. While gold grades of up to 52 g/t were produced, cleaner losses were about 18% at this grade. Further testing would be needed to improve the stage recovery of gold, and further stages of cleaning may be able to increase the grade of the concentrates as well.

13.7.4 Leaching of Flotation Tailings

A developmental leach study was performed which utilized the combined rougher + cleaner tailings from the WES locked cycle test (Gajo, 2014c). Results from tests evaluating three different feed percent solids and three different cyanide concentrations identified the optimum gold extraction from the WES tailings as 61% of the gold contained in the tailings. A batch agitated tank CIP leach was then conducted at 45% solids and 0.25 kg cyanide per tonne of tailings, which resulted in a gold extraction of 55.3%. The combined gold recovery from the locked cycle test and tailings leach for the WES composite is calculated to be 90%.

Leaching tests have also been conducted on the flotation tailings from the 29 variability samples (Table 13.18). Results from these are shown in the variability Section (13.7.7).

13.7.5 Pressure Oxidation of Concentrates and Cyanide Leaching of Residues

During the PEA phase, three pressure oxidation tests were run at Dynatec in Fort Saskatchewan, Alberta on West End gold flotation concentrates produced from the global Sub-A2 composite during that program. No regrinding of the concentrates was conducted. An acid demand test indicated that acidulation of the concentrates would be needed prior to oxidation testing, in the order of 180 kg of sulfuric acid per tonne of concentrate in testwork. The concentrate, grading 19.7 g/t Au, 9.4% S and 10.5% CO_3, was tested twice at 215°C (acidulated and with acid addition to POX) and 230°C, each with samples taken at 40, 60 and 80 minutes. Results were similar to past testing in that sulfur oxidation was very rapid with oxidation essentially complete within 40 minutes for all conditions tested (Masters, 2012).

The pressure oxidation residues were leached at McClelland Laboratories through bottle roll leaching “as is” and after regrinding to P_{80} 45 µm. The total residence time in the leach was 72 hours. The results indicated that the gold was readily leached into solution, with recoveries of 98% “as is” and 98.2% on reground residues. The silver extraction was 7.5% and 8.0%, respectively. Cyanide consumption in the tests was similar to that of Yellow Pine, at 4.3 kg/t for the “as is” leach and 4.0 kg/t for the reground residue leach, regrinding was deemed not beneficial to the leach (McClelland, 2012).

In the PFS phase of testwork, a confirmation pressure oxidation test was completed on the remaining Sub-A2 concentrate that had been used for the Dynatec PEA study (Jackman, 2014b). The pressure oxidation conditions used were recommended by Dynatec’s final report. The oxidation products were forwarded on for neutralization, CIP, CND and environmental studies.

In total, at SGS, three pressure oxidation tests were conducted at 220°C under 75 psi of O_2 for 60 minutes, including acidulation of the concentrates to pH 1.8 prior to POX. Sulfide oxidation averaged 99% in the tests with POX solutions containing 3.08 ppm arsenic and <0.5 mg/L antimony. The results supported the Dynatec autoclave recommendations, but future lab tests should confirm assumptions that acidulation is not needed with better cleaning of concentrates from West End materials.

The samples from the two PFS program pressure oxidation runs were combined, fully washed and underwent a single CIP cyanidation leach, without regrinding, to determine the extraction of gold. The gold extraction from the test was 98.3% while silver extraction was negligible; consumptions were 0.13 kg of NaCN and 10.3 kg of lime per tonne of POX solids.

13.7.6 Whole Ore Cyanidation

Thirty variability samples in total have been subjected to whole ore leaching. For the thirteen samples from the PEA phase of testwork, the kinetics were fast, generally leaching greater than 75% of the total amount in the first 5 hours of the leach. The average extraction of all transitional and oxide samples in the PEA study was 49.9%, ranging from 13.6% to 93.9%. Cyanide consumption ranged from 0.3 - 1.4 kg/t, while the lime consumption varied 0.24 - 1.5 kg/t in these tests (Ratnayake, 2013c). An additional fifteen oxide and transition samples were tested in the PFS phase to build upon that data set, showing faster yet leaching kinetics of greater than 90% in the first 5 hours. All the tests were conducted at the PEA grind size of 80 percent passing 100 microns. The average extraction of all transitional and oxide samples in the PFS study was 45.2%, ranging from 11.6% to 84.4%. Consumption of cyanide for the new samples averaged 0.45 kg/t, while the lime consumption averaged 1.37 kg/t, quite similar to the PEA sample leaches (Gajo, 2014b).

Two additional whole ore global composites, WES and West End Oxide (WEO) were also tested and achieved similar kinetic results, showing greater than 96% of the extraction happening in the first 6 hours of the leach (Gajo, 2014c). Extractions of 30% and 73% were achieved on these transitional (WES) and oxide (WEO) composites. Reagent consumption in these tests was much lower than in the bottle rolls, at just 0.02 - 0.19 kg/t for cyanide and 0.64 - 1.21 kg/t for lime.

13.7.7 Variability Testing

Variability testing has been conducted on twenty-nine West End samples spanning the entire spectrum of oxidation from true sulfide to essentially pure oxide (Ratnayake, 2013c; Gajo, 2014b). The results from flotation, flotation tailings leaching and whole ore leaching tests at 100 µm P_{80} grind are summarized in Table 13.18 (PFS samples below the double line). These results have been used for metallurgical forecasting purposes, as described later in this section. The average whole ore leach gold extraction of sixteen West End oxide samples (defined here as having less than 0.35% S in the feed) is 61.8%. The average flotation gold recovery of four sulfide samples (defined here as having greater than 0.85% S in the feed) is 81.4%. For the transition range samples, with feed sulfur grades from 0.35 to 0.85%, the best recoveries are achieved with a flotation of sulfides followed by leaching of the tailings, which averaged 89.9% total recovery of gold.

Table 13.18: Variability Testing on West End Samples

Composite ID	Head Grade (calc)			Concentrate Grade			Flotation Recovery			Total Extraction, %		
	Au (g/t)	As (%)	S (%)	Au (g/t)	As (%)	S (%)	Au (%)	As (%)	S (%)	Whole Ore Leach	Tailings Leach	Float + Tailings Leach
WE2	1.63	0.12	0.61	12.61	0.92	4.64	97.9	93.0	95.7	16.0	64.6	99.3
WE11	1.67	0.62	1.44	10.52	3.91	9.46	92.3	93.1	96.4	13.6	38.5	95.3
WE14	2.35	0.48	1.03	8.37	1.86	4.32	79.2	85.5	93.2	23.3	47.3	89.0
WE15	1.88	0.26	1.55	6.44	0.86	5.65	90.2	88.5	96.2	14.6	26.1	92.8
WE17	0.74	0.04	0.35	5.32	0.32	3.16	74.5	78.6	94.8	15.7	25.9	81.1
WE20	2.78	0.34	0.05	10.75	0.93	0.35	37.8	26.4	65.3	93.9	94.3	96.5
WE21	1.18	0.08	0.07	9.13	0.37	0.60	60.9	37.1	71.8	91.6	83.1	93.4
WE22	1.53	0.07	0.02	8.56	0.22	0.09	47.4	28.3	45.8	85.5	77.0	87.9
WE23	0.92	0.05	0.01	6.05	0.19	0.07	44.2	25.5	34.5	80.0	74.4	85.7
WE24/25	1.25	0.13	0.33	8.09	0.71	3.06	66.9	56.4	94.6	46.0	85.7	95.3

Composite ID	Head Grade (calc)			Concentrate Grade			Flotation Recovery			Total Extraction, %		
	Au (g/t)	As (%)	S (%)	Au (g/t)	As (%)	S (%)	Au (%)	As (%)	S (%)	Whole Ore Leach	Tailings Leach	Float + Tailings Leach
WE26	0.64	0.03	0.10	3.67	0.18	0.92	53.4	51.4	82.5	68.7	84.5	92.8
WE27	0.63	0.03	0.16	3.90	0.22	1.46	57.1	63.3	83.3	56.1	76.1	89.7
WE28	1.72	0.05	0.10	20.00	0.50	1.77	50.7	45.8	80.4	66.5	69.7	85.1
WE29	0.52	0.07	0.32	6.90	1.00	4.91	82.0	81.9	94.2	11.3	8.6	83.5
WE30	0.30	0.11	0.18	1.90	0.50	3.54	27.2	20.1	84.2	84.4	91.5	93.8
WE33	0.37	0.07	0.17	2.70	0.40	1.55	75.4	66.8	89.6	29.0	40.0	85.3
WE34	0.82	0.09	0.14	7.10	0.70	1.70	59.2	51.9	80.7	52.8	69.9	87.7
WE35	0.64	0.22	0.55	2.30	0.90	3.46	49.1	54.4	87.4	78.0	89.6	94.7
WE38	1.39	0.21	0.17	8.60	1.20	1.94	48.3	44.5	89.1	72.7	89.9	94.8
WE39	1.45	0.17	0.31	16.50	2.30	4.50	73.5	85.3	93.9	46.5	64.8	90.7
WE40	0.76	0.13	0.39	13.60	2.40	7.74	82.5	82.9	92.6	14.8	0.0	82.5
WE41	3.16	0.31	0.89	17.10	1.70	6.63	63.7	65.8	87.2	37.2	61.9	86.2
WE42	1.53	0.24	0.57	13.80	2.20	5.66	85.2	84.3	93.7	17.2	24.5	88.9
WE43	1.68	0.18	0.41	34.50	4.10	11.10	66.7	74.9	88.2	36.9	71.6	90.5
WE44	1.70	0.11	0.41	31.10	2.60	12.30	55.6	70.1	90.6	57.1	75.7	89.2
WE45	1.34	0.03	0.42	17.20	0.40	7.37	65.3	55.8	88.8	43.8	79.1	92.8
WE47	0.90	0.11	0.29	23.00	2.90	7.96	82.9	84.7	90.0	29.4	44.7	90.5
WEO	0.91	n/a	0.04	n/a	n/a	n/a	n/a	n/a	n/a	73.6	n/a	n/a
WES	1.62	0.24	0.67	10.00	1.50	5.10	79.7	83.7	93.4	29.7	55.3	90.9

The actual use of leach, flotation and flotation in combination with tailings leach would be driven by project economics and thus, the cut-off grades of sulfur for samples processed in each manner may change.

13.8 HISTORIC TAILINGS REPROCESSING

Approximately 2.7 million tonnes of historic tailings, produced and deposited during the 1930s through the 1950s by the Bradley Mining Company, are located in the Meadow Creek Valley. These tailings average approximately 1.19 g/t gold, 2.92 g/t silver and 0.17% antimony; consequently, a test program was completed to assess whether processing them is economically feasibility; based on the information currently available, no testwork has been completed in the past evaluating the reprocessing of the tailings. The testwork program considered blending the tailings with early production Yellow Pine material as that timing coincides with when they would have to be processed prior to development of the waste rock storage facility that is planned for that area.

Particle size analyses of the seven composites tested show an average P_{80} of 193 µm with a range from 109 µm to 323 µm and a gold head grade ranging from 0.78 g/t to 1.51 g/t.

Table 13.19: Head Grade and Particle Size Analyses of Historic Tailings Composites

Head Grade	Comp 1 (24S)	Comp 2 (25S)	Comp 3 (26S)	HTL	HTM	HTH	S06
Au, g/t	0.98	1.12	1.51	0.78	1.17	1.31	1.44
Ag, g/t	4.00	3.00	3.80	-	-	-	< 10
As, %	0.09	0.15	0.18	0.09	0.15	0.17	0.10
Sb, %	0.14	0.16	0.22	0.07	0.23	0.17	0.87
S, %	0.43	0.36	0.29	0.18	0.37	0.28	0.61
PSA P_{80}, µm	323	142	109	139	276	116	245

Testwork completed where the historic tailings material was blended into the early production Yellow Pine material at a ratio of 15% of the total feed to the ball mill used single point calibration data to determine relative work indices and laboratory grind times. This scoping level data indicates that blending the historic tailings material with fresh ore may reduce the operating work index of the total feed to the grinding circuit by 10 - 14% (Gajo, 2014b).

13.8.1 Flotation

Two courses of testwork have been undertaken at SGS on the Historic Tailings, the first being a scoping study into the response of the tailings to the standard flotation procedure that was conducted on the oxide-transitional materials of the three primary ore deposits (McCarley, 2013). The second study applied the Yellow Pine flotation flowsheets for low antimony and high antimony material to four different composites of the historic tailings (McCarley, 2014), which culminated in a locked cycle testwork program to evaluate the effect of a 15% blend of historic tailings with fresh, Early Production Yellow Pine material on flotation response (Gajo, 2014b).

The scoping study was conducted on three single sonic core composites (Comp 1, Comp 2 and Comp 3) which were spatially separated in the repository. Rougher flotation was conducted at 100 µm and 75 µm P_{80} grind targets, at natural pH, with 200 g/t copper sulfate as the activator, 110 g/t PAX as the collector and froth collection of 29 minutes. An additional set of tests, done at 75 µm grind, included sodium sulfide (Na_2S) in a conditioning step after the grind. Gold recovery in all samples was improved with the finer 75 µm P_{80} target in the grind over the 100 µm target while sulfidizing with Na_2S improved the performance of flotation for Composites 1 and 2 but had no discernible effect on Composite 3. The best performance achieved with the 75µm tests and no sulfidizing were:

- Composite 1: Concentrate assaying 5.23 g/t Au and 2.1% S at 84.3% gold recovery;
- Composite 2: Concentrate assaying 4.26 g/t Au and 1.4% S at 78.7% gold recovery; and
- Composite 3: Concentrate assaying 2.62 g/t Au and 0.45% S concentrate at 64.2% gold recovery.

Rougher mass pulls ranged from 17 to 37%. Kinetic cleaner testing was then conducted with 10 g/t PAX. The cleaner stage gold recoveries obtained in the tests without sodium sulfide were: Composite 1 with 94.4%, Composite 2 with 77.3% and Composite 3 with 85.5%.

The second flotation program for historic tailings evaluated composites which had been chosen considering the spatial relationship of the core holes in the repository, and also by gold grade. The composites are Historic Tailings High gold, 1.31 g/t (HTH), Historic Tailings Average gold, 1.17 g/t (HTM), and Historic Tailings Low gold, 0.78 g/t (HTL). An additional, single-hole variability composite (composite # S06) was also tested as a high antimony sample (0.87% Sb). The purpose of the testwork was to gauge the response of the tailings to the established Yellow Pine flotation flowsheet. Later testwork in the PFS study included flotation of fresh Yellow Pine ore samples blended with the HTM composite and evaluation of the flotation response.

The composites were subjected to kinetic rougher flotation testing on either the Yellow Pine High antimony or Low antimony flowsheet, based on their antimony content. Antimony circuit antimony recovery from the high-antimony composites HTM, HTH, and S06 were 37%, 12% and 49%, respectively. Gold losses to the rougher concentrate were all below 3.2%. The overall results of the kinetic tests for each composite were:

- HTL at 6.86 g/t Au, 1.6% S and 54.5% gold recovery;
- HTM with 15.9 g/t Au, 5.42% S and 71.1% gold recovery;
- HTH at 2.41 g/t Au, 0.7% S and 36.6% gold recovery; and
- S06 at 14.1 g/t Au, 6.2% S and 75% gold recovery.

The kinetics data suggest gold flotation was near completion for the HTM and S06 composites using the standard residence time, but flotation was slow in HTL and HTH and not complete (the latter was reported to be very viscous in the cell).

Cleaner tests were then conducted on the composites to evaluate the effect of the Yellow Pine flowsheet on cleaning of the historic tailings. Composite HTH was floated at 30% solids rather than at 35% and it was noted that it was still quite thick but did not appear to be as rich in slimes. It is felt that when blended into fresh ore, this influence on viscosity will not be observed in flotation, but further testwork is recommended to confirm such. A summary of the cleaner flotation test results are given in Table 13.20.

Table 13.20: Historic Tailings Cleaner Test Results with Yellow Pine Flowsheet

Parameter	HTL	HTM	HTH	S06	Parameter	HTL	HTM	HTH	S06
Feed P_{80}	81	71	61	68	-	-	-	-	-
Antimony 2nd Cleaner Concentrate					*Gold Final Concentrate*				
Mass Pull (%)	-	0.13	0.08	0.17	Mass Pull (%)	3.89	2.81	2.83	2.73
Grade					Grade				
Au, g/t	-	7.62	6.01	3.30	Au, g/t	11.5	25.7	12.4	32.9
As, %	-	0.35	0.60	0.18	As, %	1.21	2.98	1.70	3.07
Sb, %	-	50.4	6.79	60.2	Sb, %	-	1.44	1.48	10.4
S, %	-	20.7	3.67	24.8	S, %	2.85	8.94	4.47	14.8
Recovery					Recovery				
Au, %	-	0.94	0.53	0.45	Au, %	55.4	68.7	36.6	70.8
As, %	-	0.30	0.30	0.24	As, %	51.8	55.3	28.0	64.1
Sb, %	-	27.5	3.44	13.3	Sb, %	-	17.0	24.9	35.9
S, %	-	7.16	1.16	6.96	S, %	61.1	67.0	47.3	64.9

Both HTM and S06 yielded antimony concentrates that were at or above the 50% grade target for antimony content; all three antimony final concentrates held the gold losses to less than 1% but antimony recovery was poor in all three tests. In the case of S06 this resulted in a high antimony grade of nearly 10.5% in the final gold concentrate, while in the other tests the antimony remained low, below 1.5%. Gold concentrates from both HTM and S06 were above 5% sulfur in rougher flotation. Since the historic tailings are proposed to be blended into the Yellow Pine mill feed at a 15% ratio, this is not expected to have a significant impact on the overall concentrate sulfur grade produced from the blend.

13.8.2 Leaching Studies

All seven historical tailings composites were leached by a standard cyanidation procedure to determine the overall recovery of gold from whole samples. In the scoping program on the first three composites, one set of tests was completed at a grind P_{80} target of 100 µm, in line with the oxide-transition testwork. A second set of tests was completed at the same grind target and using oxygen addition to the leaches. A third set of tests was then completed at the new PFS grind P_{80} target of 75 µm with oxygen. For the four developmental program composites, a single set of leaches was conducted at the P_{80} target of 75 µm with oxygen.

The results of the leaches on the first three composites in the scoping program mirrored the flotation results in that Composite 3 was now best performing (poorest flotation), followed by Composite 2 and then finally Composite 1 (best flotation). Neither the use of oxygen nor the finer grind appeared to benefit the extraction of gold from any of the samples. Leach kinetics showed that most of the gold was leached within the first 24 hours of the test.

In the developmental study leaches, the extractions of gold from the feed also mirror the flotation results; where HTL floated 55 - 60% of the gold, approximately 36% leached, indicating that much of what did not float is associated with non-sulfide gangue minerals. There is a similar pattern for HTM, with 73 - 77% floating and 28% leaching, in HTH, with 58 - 69% floating and 49% leaching and in S06, with 79 - 83% floating and 26% leaching.

Select tailings from the flotation tests were also subjected to the standard cyanidation leaches, with the results given in Table 13.21. For tests HTM and S06 the tailings leach was conducted on rougher tailings alone, while for HTL and HTH the leaches were conducted on a combination of rougher and cleaner tailings.

Table 13.21: Flotation Tailings Leach Results on Historic Tailings Composites

Comp ID	P_{80} (µm)	Average Dissolved O_2 (mg/L)	Reagent Consumed		Gold Head Grade		Gold Residue (g/t)	Gold Extraction	
			NaCN (kg/t)	Lime (kg/t)	Calc (g/t)	Direct (g/t)		(%)	(g/t)
Comp 1	92.0	33.7	0.35	0.83	0.25	0.25	0.19	26.0	0.06
Comp 2	81.0	28.6	0.31	0.86	0.38	0.29	0.24	37.2	0.05
Comp 3	77.0	31.5	0.51	1.89	1.09	1.07	0.47	57.0	0.60
HTL	81.0	26.6	0.20	0.61	0.34	0.37(1)	0.22	35.6	0.15
HTM	71.0	23.9	0.16	0.54	0.28	0.28	0.23	16.5	0.05
HTH	61.0	13.7	0.77	0.86	0.59	0.58(1)	0.41	31.7	0.17
S06	68.0	25.8	0.23	0.64	0.27	0.27	0.21	22.4	0.06
Note: (1) Estimated from Rougher + Cleaner tailings assays									

When combined with the amount of gold recovered through flotation, the overall gold recovery of the historic tailings becomes 84.3%, 87.6% and 73.7% for Composites 1, 2 and 3, respectively. It becomes 71.3%, 75%, 56.7% and 80.6% for Composites HTL, HTM, HTH and S06, respectively. Including gold recovered to the antimony circuit, HTM, HTH and S06 overall gold recovery becomes 78.2%, 60.8% and 81% respectively.

Testwork was conducted on blends of Yellow Pine fresh material and Historic Tailings, with the aim of exploring if the two feed types could be effectively co-mingled for processing. This testwork included a brief batch test program on the EPH and EPL composites, each time blended with 15% of average gold grade (HTM2) Historic Tailings material (Gajo, 2014b). This culminated in a locked cycle test on the blended EPH/Historic Tailings composite. As the EPL/Historic Tailings blend reached target sulfur grade by roughing alone, this was not tested in locked cycle mode, rather by a 4 kg batch test. The results, shown below, indicate no adverse effects of blending in the tailings on overall metallurgy, with both antimony and gold recoveries very similar to those from testing Yellow Pine alone, and antimony concentrate grades remaining very high and sulfur grades remaining above the threshold for POX.

Table 13.22: Flotation of Blended Yellow Pine Early Production Feed and Historic Tailings

Material	Weight		Assays					Distribution			
	Dry	%	Au (g/t)	As (%)	Sb (%)	S (%)	CO_3 (%)	Au (%)	As (%)	Sb (%)	S (%)
Blend of Early Production High Sb (85%) & Historic Tailings (15%)											
LCT Sb Final Concentrate	17.2	0.86	11.4	0.39	58.7	25.8	n/a	3.3	0.8	89.1	15.9
LCT Au Rougher Concentrate	354.7	17.7	15.1	2.09	0.28	6.3	1.7	91.3	92.2	8.9	80.9
LCT Au Rougher Tail	1635.4	81.5	0.19	0.03	0.01	0.05	0.85	5.4	7.0	1.9	3.1
Blend of Early Production Low Sb (85%) & Historic Tailings (15%)											
BT Au Rougher Concentrate	633.4	15.8	12.4	2.17	n/a	5.98	1.45	94.3	93.4	n/a	95.7
BT Au Rougher Tail	3367.7	84.2	0.14	0.03	n/a	0.05	n/a	5.7	6.6	n/a	4.3
Note: LCT - Locked Cycle Test, BT - Batch Test											

The tailings were leached using the standard Yellow Pine leach procedure, with 8% of the gold in the EPH Blend flotation tailings and 20% of the gold in the EPL Blend tailings being leached – the latter being slightly better than might have been expected from a pure weighted average of the individual metallurgy of the two components.

Overall, the above evidence suggests that the Yellow Pine and Historic Tailings materials can be successfully co-processed.

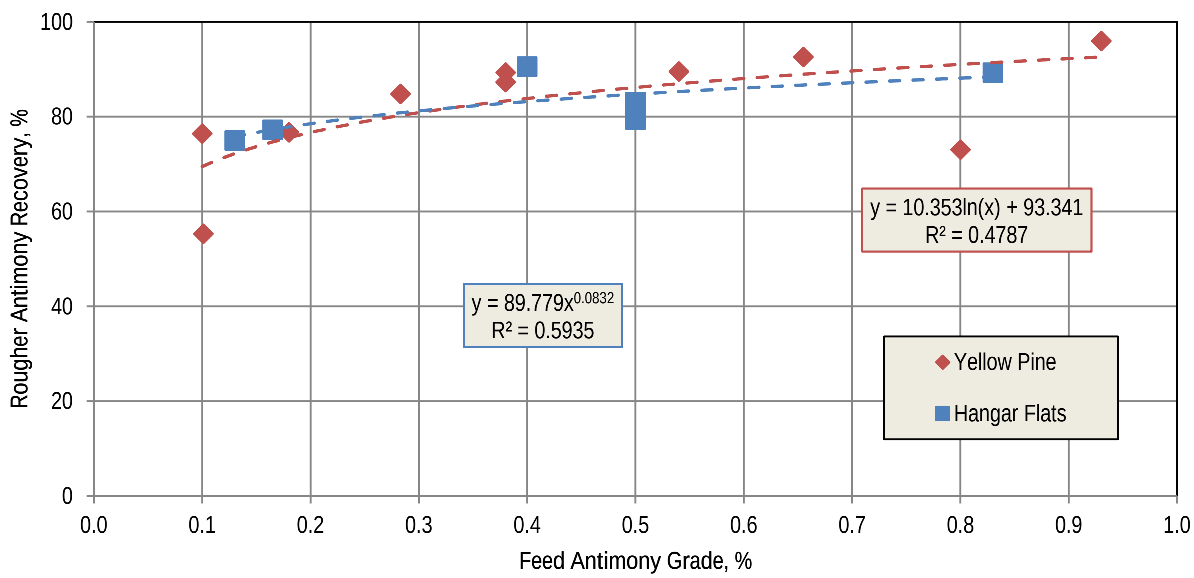
13.9 METALLURGICAL PREDICTION

13.9.1 Antimony Flotation

The PFS metallurgical testing program employed rougher flotation tests to describe the variability in antimony recovery, with the use of a small number of locked cycle tests to describe how well the antimony floated to the rougher concentrate and was upgraded by closed-circuit (locked cycle) cleaner flotation to saleable grade.

The rougher flotation results have been tabulated in previous sections as Table 13.5 and Table 13.11. Antimony recovery is linked to head grade for both Yellow Pine and Hangar Flats as shown on Figure 13.5, and equations linking head grade to a prediction of recovery are shown for both Yellow Pine and Hangar Flats.

Figure 13.5: Antimony Rougher Metallurgical Prediction



Three antimony flotation locked cycle tests were completed, the results from which have been described earlier in this section and are summarized below. Cleaner circuit performance is relatively consistent between the three tests yielding a concentrate assaying 59% Sb at roughly 96% cleaner stage recovery. Accordingly, antimony cleaner recoveries of 98.1% (the mean of the two tests) for Yellow Pine and 96.7% for Hangar Flats has been applied to the Sb rougher recoveries from the above equations to link antimony feed grade to recovery to final concentrate. A constant concentrate grade of 59% Sb has been assumed.

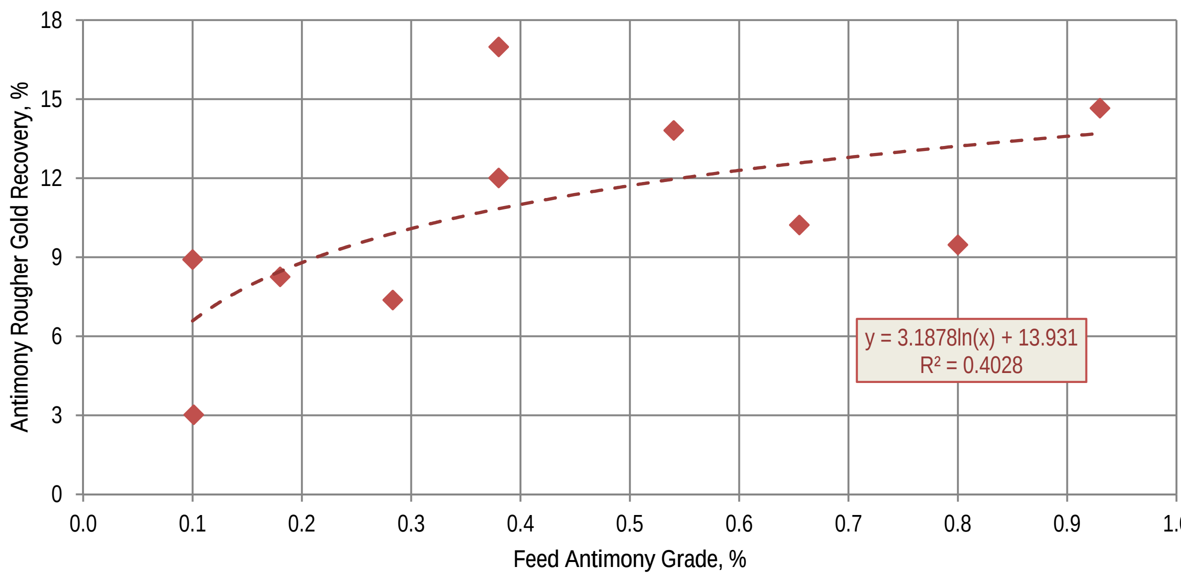
13.9.2 Flotation and Direct Leaching of Gold

13.9.2.1 Yellow Pine

The recovery of gold to the combined Yellow Pine rougher concentrates has been characterized for both the antimony-rich and antimony-poor samples using the data from sixteen variability tests conducted on early production samples, samples from the 2014 mini-variability program and on the major PFS composites (see Table 13.1).

While the use of the sequential antimony-gold flotation circuit has no adverse effect on overall gold recovery, the gold recovered to the gold concentrate itself is lower due to misplacement to the antimony final concentrate. In Yellow Pine, this is linked to the antimony head grade.

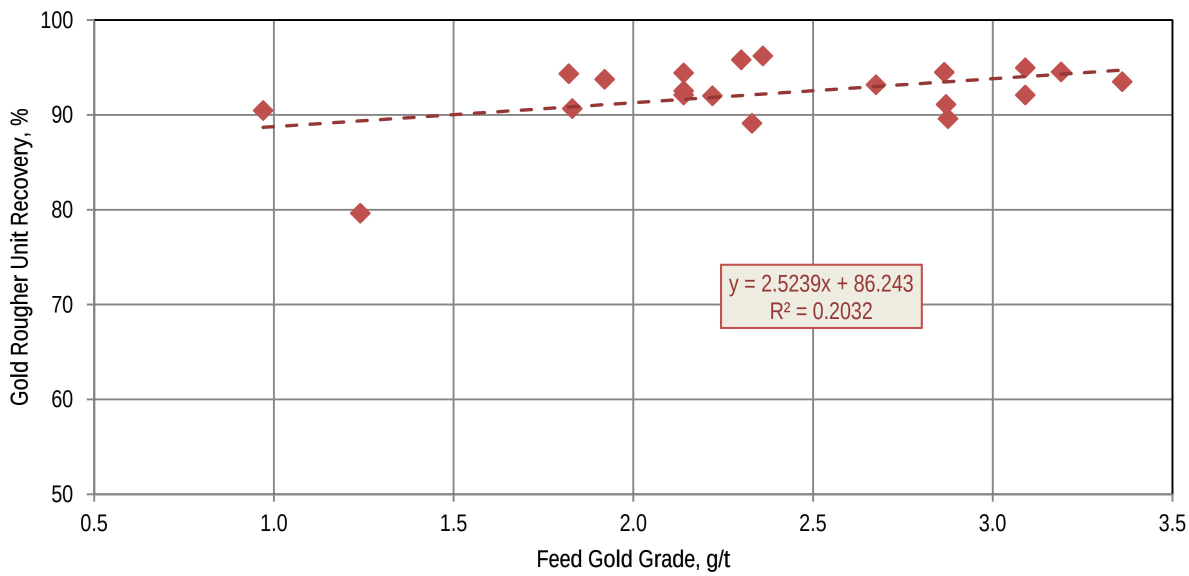
Figure 13.6: Yellow Pine Antimony Head Grade *vs* Antimony Rougher Gold Loss



In the final prediction equation, this value must be multiplied by a factor of 0.237 to account for the rejection of some of the rougher gold in cleaning that was observed in locked cycle flotation.

Unlike in the PEA, the PFS-optimized flowsheets demonstrated no systematic differential in overall gold recovery with the use of the two flowsheets, so all the rougher flotation data has been gathered into a single dataset. There is evidence, albeit weak, of a gold head grade/rougher recovery relationship so a simple linear regression has been used in preference to using a fixed recovery.

Figure 13.7: Yellow Pine Gold Head Grade *vs* Rougher Unit Recovery



The regression chart and equation shown on Figure 13.7 provide the flotation recovery of gold in samples with a feed grade of less than 4 g/t gold as a function of head grade.

Sulfur recovery appears fixed and independent of head grade, averaging 94.7%, and in the low antimony samples the concentrate sulfur grade also appears to be independent (Figure 13.8). In the high antimony samples there is a trend favoring higher concentrate grades as a consequence of higher head grades, and this has been used to predict the concentrate sulfur grade to the concentrate in those samples (Figure 13.9).

Figure 13.8: Yellow Pine Sulfur Head Grade *vs* Rougher Concentrate Recovery and Grade

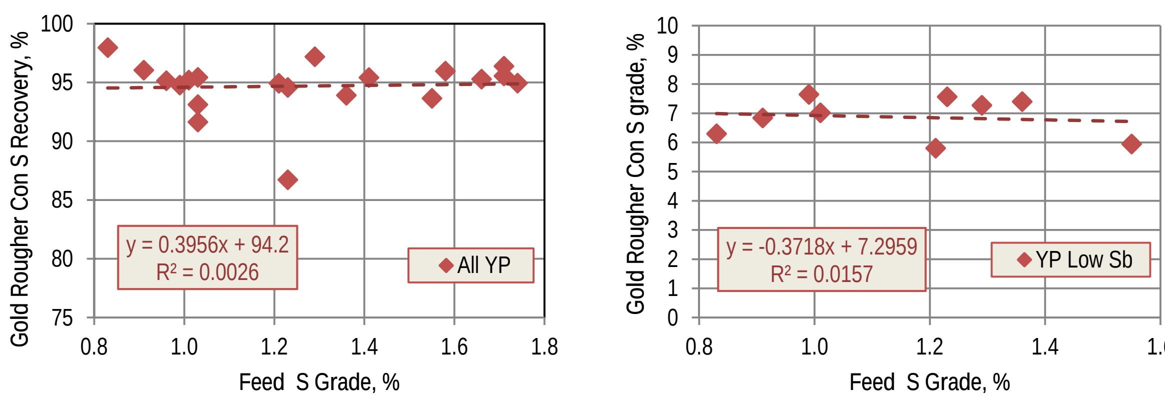
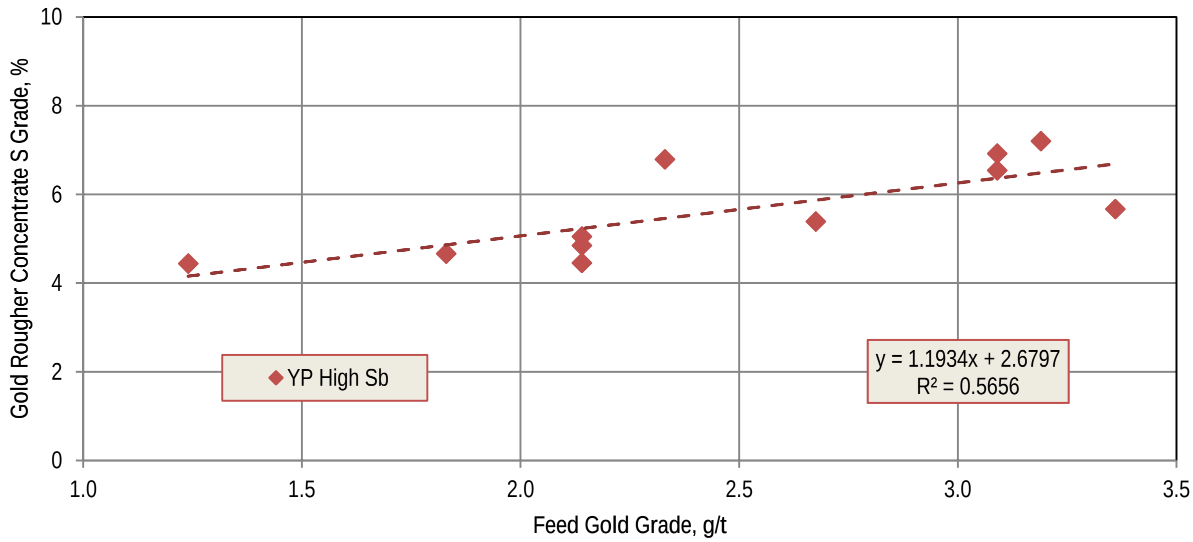


Figure 13.9: Yellow Pine High Antimony Sample Gold Head Grade *vs* Rougher Concentrate Sulfur Grade



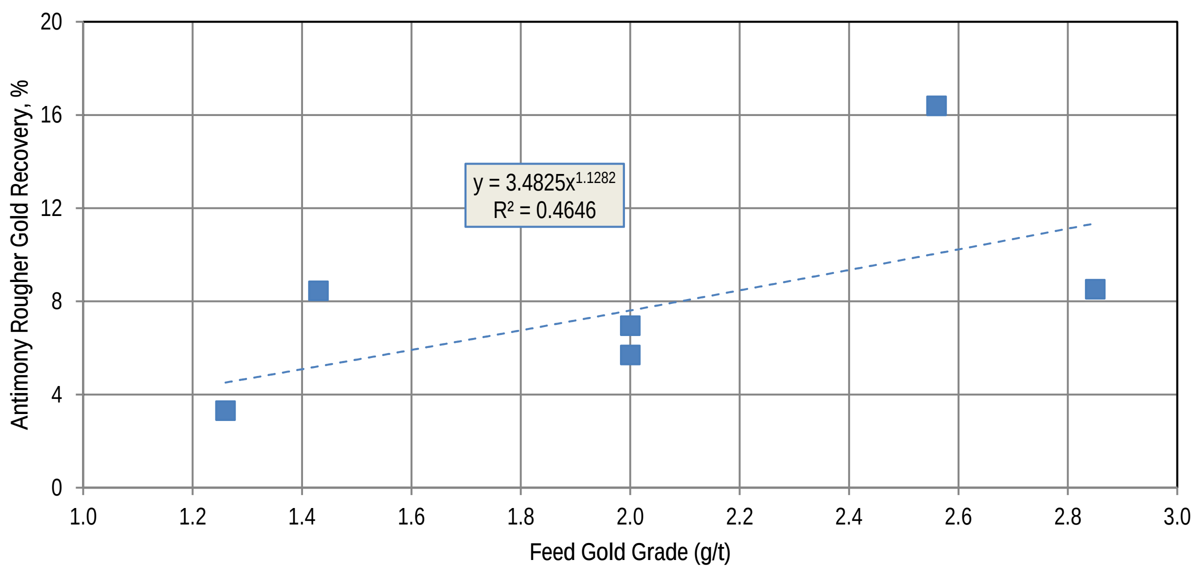
Based on the limited data available on the carbonate assay in the flotation concentrates, a factor of 0.63 x head carbonate grade has been used to predict the carbonate grade in the Yellow Pine gold concentrates.

Leaching the tailings yielded generally very poor recoveries of additional gold. A constant leach extraction reflecting the average of the dataset (10% of the gold in the tailings) has been assumed for metallurgical forecasting purposes.

13.9.2.2 Hangar Flats

In Hangar Flats, gold misplacement to the antimony rougher concentrate is best predicted using the feed gold grade.

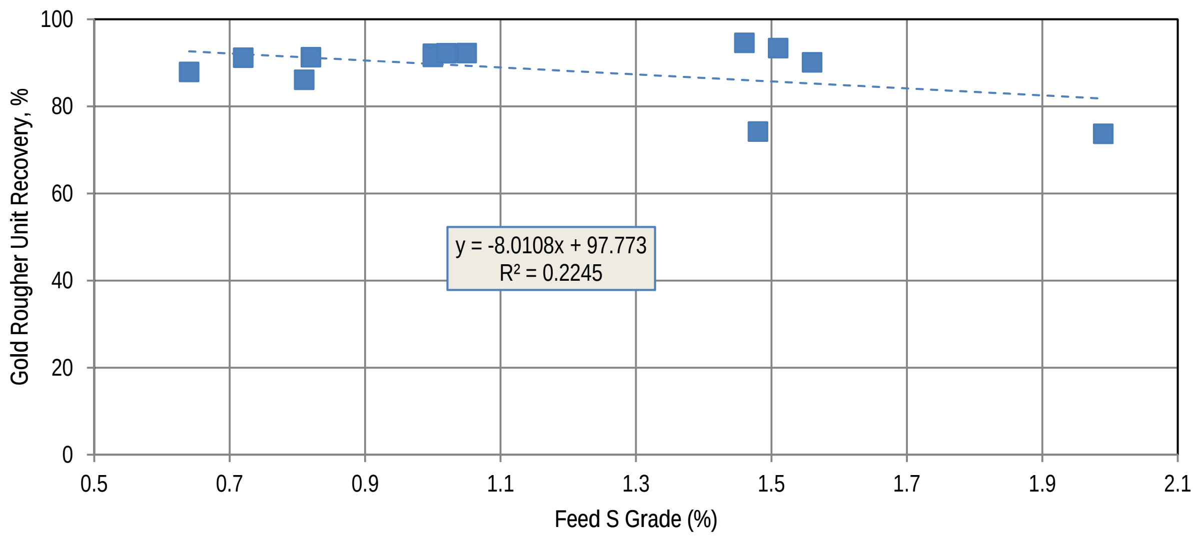
Figure 13.10: Hangar Flats Gold Head Grade *vs* Antimony Rougher Gold Loss



In the final prediction equation, this value must be multiplied by a factor of 0.239 to account for the rejection of some of the rougher gold in cleaning that was observed in locked cycle flotation.

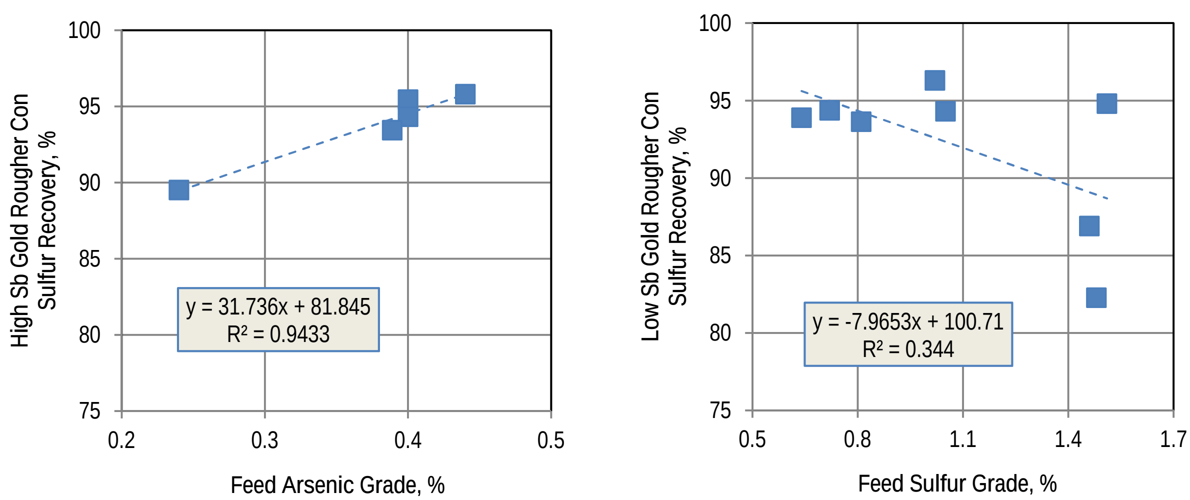
As with Yellow Pine, gold recovery from sulfide Hangar Flats samples appeared to be unrelated to the use of the bulk flotation or Sb-Au sequential flowsheet. A sulfur head grade/gold rougher recovery relationship has been used to predict gold recovery in preference to using a fixed recovery (for all low and high antimony samples). Although this negative correlation is somewhat counterintuitive, the limited data available meant that the two poor acting samples could not be ignored and the relationship has accordingly been adopted.

Figure 13.11: Hangar Flats Feed Sulfur Head Grade *vs* Gold Rougher Flotation Gold Recovery



Lower recoveries can be expected with gouge samples influenced by the Meadow Creek Fault, as evidenced by the two low performers in the chart, which are expected to make up approximately 15% of the Hangar Flats Mineral Resource. It may be beneficial in the future to segregate out the projections for gouge-influenced samples.

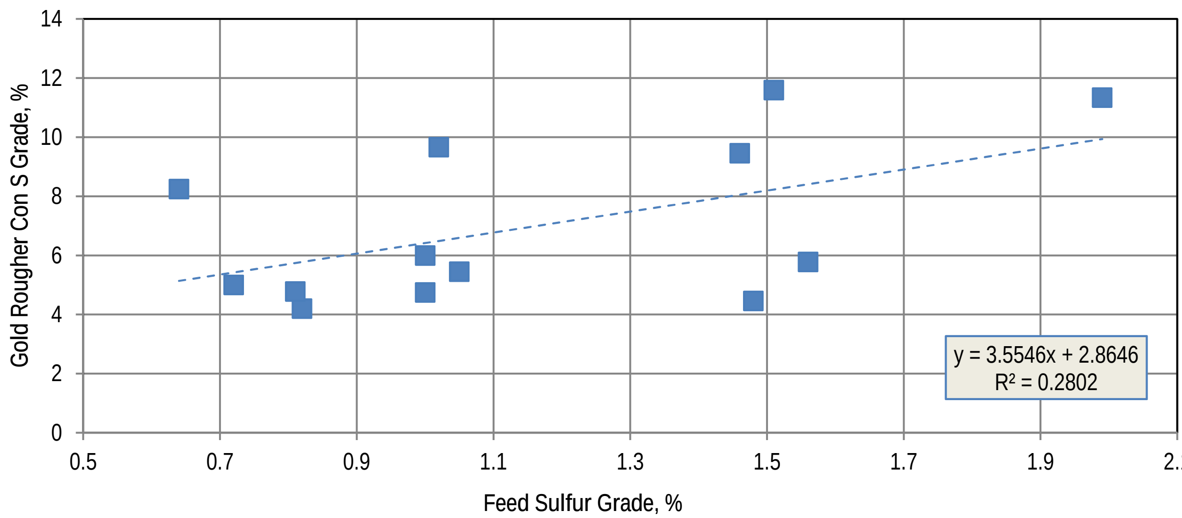
Figure 13.12: Hangar Flats Gold Rougher Concentrate Sulfur Recovery Prediction



Sulfur recovery to the gold concentrate in High Sb samples appears strongly tied to the feed arsenic grade while for the Low Sb samples it is more closely tied to feed sulfur grade (Figure 13.12). The influence of the gouge material is again seen with lower recovery in higher sulfur samples.

Sulfur grade in the rougher concentrates of the full suite of samples is best modeled with the feed sulfur grade as shown on Figure 13.13.

Figure 13.13: Hangar Flats Sulfur Head Grade *vs* Rougher Sulfur Grade

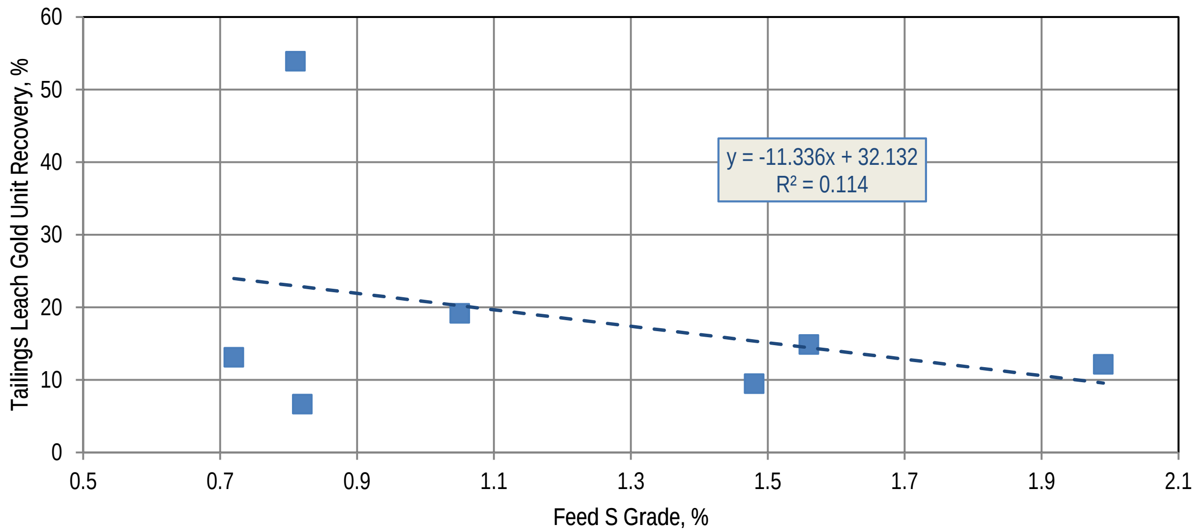


Based on the limited data available on the carbonate assay in the flotation concentrates, a factor of 0.74 x head carbonate grade has been used to predict the carbonate grade in the Hangar Flats gold rougher concentrates.

In the final years of operation, the planned mill feed is dominated by West End material with a minor component of material from Hangar Flats. The processing of the carbonate-rich West End material would prompt the need to clean the rougher concentrates to achieve the target 0.9:1 ratio of carbonate to sulfur. There would also be occasional times where the Hangar Flats concentrate sulfur grade would not meet the 5% sulfur requirement for POX (gouge material and some transitional ores). The effect of cleaning on overall grades and recoveries has been established by comparing the recoveries by locked cycle testing of the HFH and HFL with commensurate rougher testing. The HFH and HFL locked cycle concentrates assayed 14.5% and 14.9% sulfur respectively, at cleaner stage gold recoveries of 95.3% and 96.2%. For the sake of mass balancing these have been assumed to be constant at the average (95.8%) gold recovery and (14.7%) sulfur grade through the years when cleaner flotation would be operating. Sulfur recovery to the cleaner concentrate is also assumed fixed at 96.1%. Based on limited data, carbonate grade does not appear to change with cleaning and should be assumed to remain the same as measured in the rougher concentrate.

The leach response from the Hangar Flats flotation tailings has not been well-established in the test program – tailings from some poor-floating Hangar Flats samples leached well (the transition samples) and the HFH and HFL composites both yielded economic leach recoveries from their tailings while some other poor-floating samples leached poorly. A loose relationship exists however, between feed sulfur grade and the recovery of gold from the tailings.

Figure 13.14: Hangar Flats Feed Sulfur Grade *vs* Flotation Tailings Gold Leach Unit Recovery



13.9.2.3 West End

Two flowsheets would be employed to treat West End ores: cleaner flotation plus tailings leaching, and direct whole ore cyanidation (i.e. bypassing the flotation circuit). Process selection would be driven by the differential in recovery from the two options traded against the processing cost differential.

Gold flotation recoveries are highest when the gold is in solid solution in the host sulfides, and at their lowest when the sulfides have been totally destroyed by oxidation. Owing to its grain size, this free gold in oxide samples does not float well. Accordingly, the PEA variability sample flotation recovery shows an inverse correlation with the geochemical leach recovery.

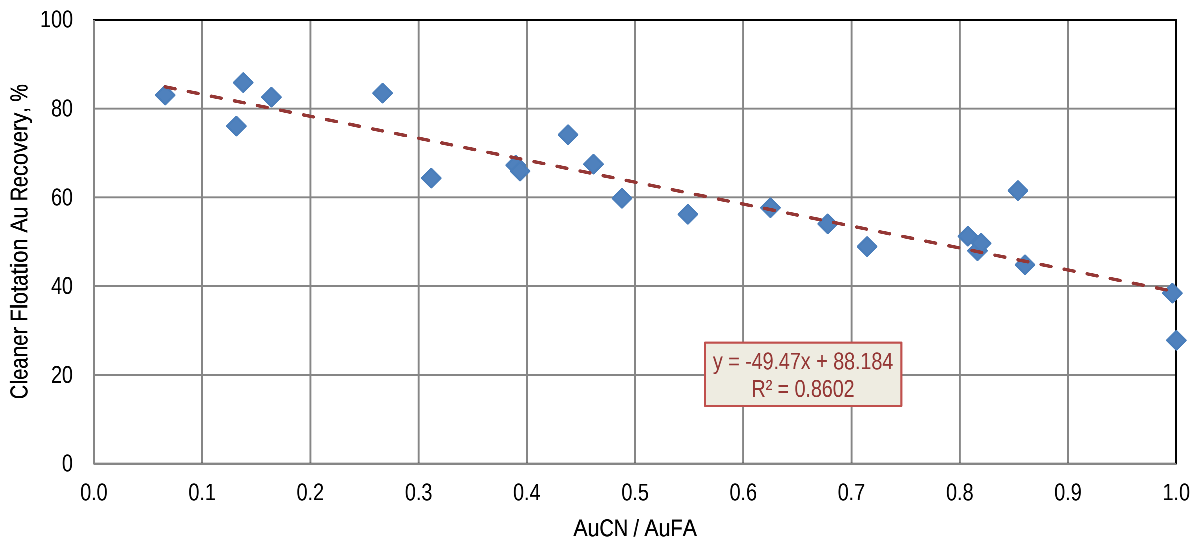
The most recent PFS variability testwork was conducted on the optimized flowsheets at a primary grind P_{80} of 75 µm, and as compared to the PEA primary grind of P_{80} at 100 µm, showed improved flotation and tailings leach recoveries which needed to be accounted for in the current PFS metallurgical projections. PFS cleaning of the rougher concentrates at this finer grind was able to produce a POX ready flotation concentrate at slightly improved recovery as compared to the PEA rougher concentrate recovery. The mean difference in recoveries between the PEA rougher and PFS cleaner flowsheets for flotation is shown below:

Table 13.23: Comparison of PEA and PFS Flowsheet Flotation Recoveries to POX Feed

Sample	PEA (%)	PFS (%)	Difference (%)
WE41	63.7	62.8	-0.9
WE44	55.6	56.7	1.1
WE47	82.9	84.5	1.6
Average	67.4	68.0	0.6

It must be stressed that the above 0.6% improvement in recovery is directly related to cleaned POX-ready concentrate, and that no account therefore needs to be made for cleaner losses in using this number to convert from the AuCN ratio-based equations to POX ready concentrate. This negates the PEA projection requirement for a 0.975 cleaning performance factor and is incorporated into the equation as shown on Figure 13.15.

Figure 13.15: West End Flotation Gold Recovery *vs* Whole Ore Geochemical Leach Gold Extraction



The sulfur grades of the cleaner concentrates from three batch tests and one locked cycle test on WE41, WE44, WE47 and WES respectively were 12.5%, 12.3%, 13.1% and 14.4%, despite highly variable sulfide sulfur head grades of 0.91%, 0.41%, 0.26% and 0.68% respectively. Accordingly, it has been assumed that the sulfur grade in the concentrate would be a constant, and the average of the tests (13.0%) has been used. The mass pull has been calculated by balancing the sulfur grades in the feed and concentrate, together with the sulfur recovery – which is assumed to be 93% based on the locked cycle test (the mean batch sulfur recoveries from the batch tests averaged 94%).

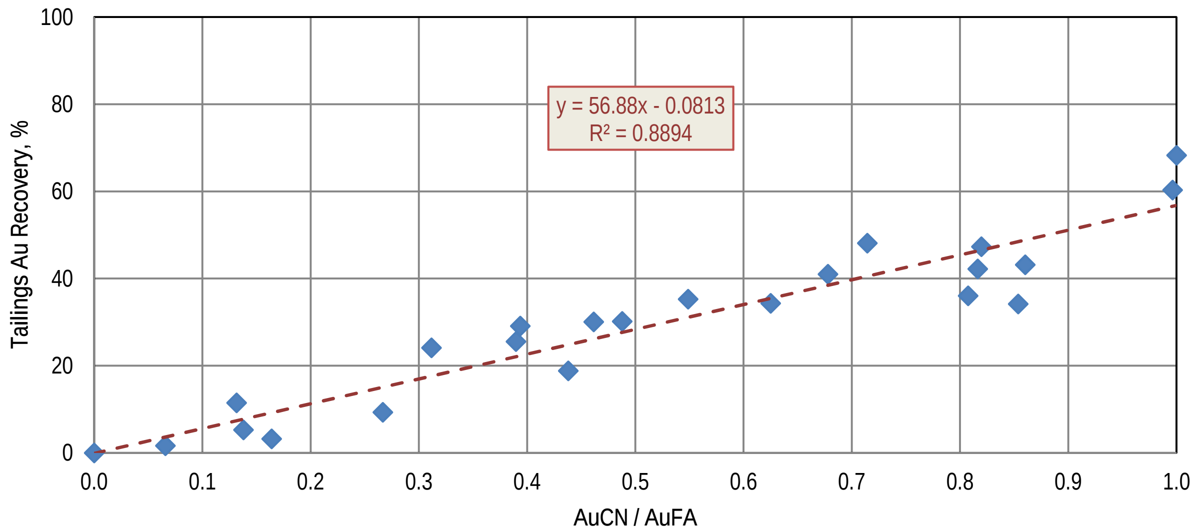
Tailings from the float are for the most part leachable, and correlate well with the geochemical gold cyanide (AuCN) / gold fire (AuFA) assays. Due to the finer primary grind size in the PFS flowsheet, now at 75 µm instead of 100 µm, the leachability of the gold in tailings has improved. The mean difference in recoveries between the PEA and PFS flowsheets for tailings leaching is shown in Table 13.24.

Table 13.24: Comparison of PEA and PFS Flowsheet Flotation Tailings Leach Gold Extraction

Sample	PEA (%)	PFS (%)	Difference (%)
WE41	22.5	25.6	3.1
WE44	33.6	33.5	-0.1
WE47	7.6	9.6	2.0
Average	21.2	22.9	1.7

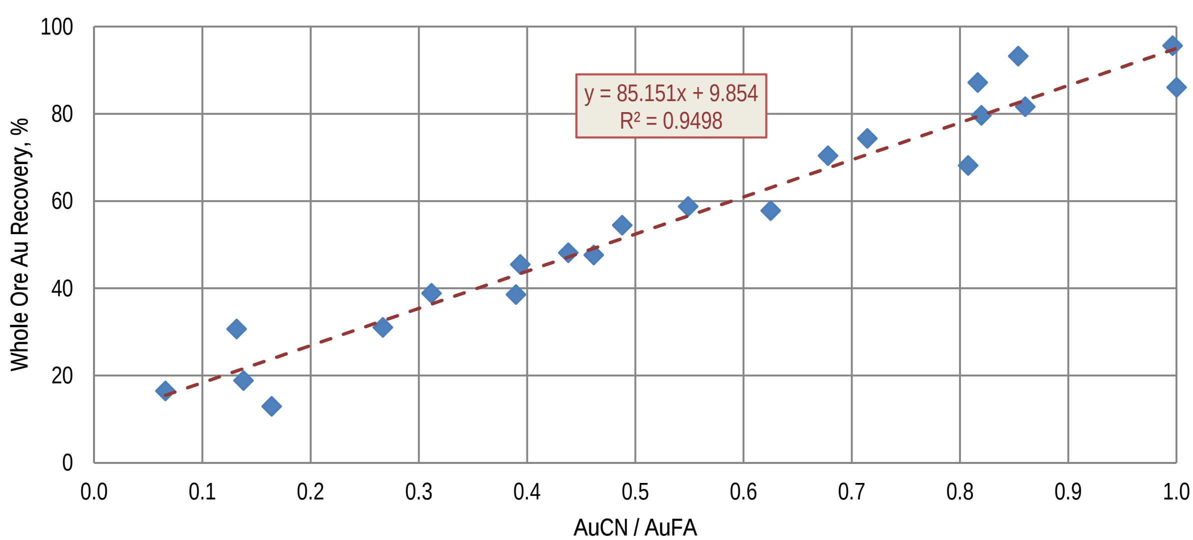
This data analyzed for the prediction equation includes the point (0, 0); it is felt that it is a legitimate point since zero extraction from the AuCN tests would indicate zero extraction from a laboratory leach of tailings. Also, as noted above, moving from the 100 µm PEA flowsheet to the 75 µm PFS flowsheet improves the leach recovery by an average of 1.7%, which is also incorporated into the analysis. The respective chart and equation developed is shown on Figure 13.16. It has been assumed for the sake of metallurgical projections that the gold lost in cleaning will subsequently leach to the same recovery as the gold in the rougher tailings.

Figure 13.16: West End Flotation Tailings Leach *vs* Whole Ore Geochemical Leach Gold Extraction



The extraction of gold from whole ore leaching is best defined through the modeled cyanide gold assay (AuCN) diagnostic geochemical data, adjusted to fit expected actual commercial leaching performance. The AuCN assays are compared with AuFA assays to evaluate the maximum percentage of gold that is leachable. This short, intensive, cold cyanide leach on material of a nominal 100 µm grind (but in reality quite pulverized), tended to yield recoveries somewhat higher than metallurgical laboratory bottle roll data used in the regression study (Table 13.18). The equation on Figure 13.17 was developed to arrive at a link between the geochemical and metallurgical laboratory recoveries.

Figure 13.17: West End Metallurgical Laboratory Whole Ore Leach *vs* Geochemical Leach Gold Extraction

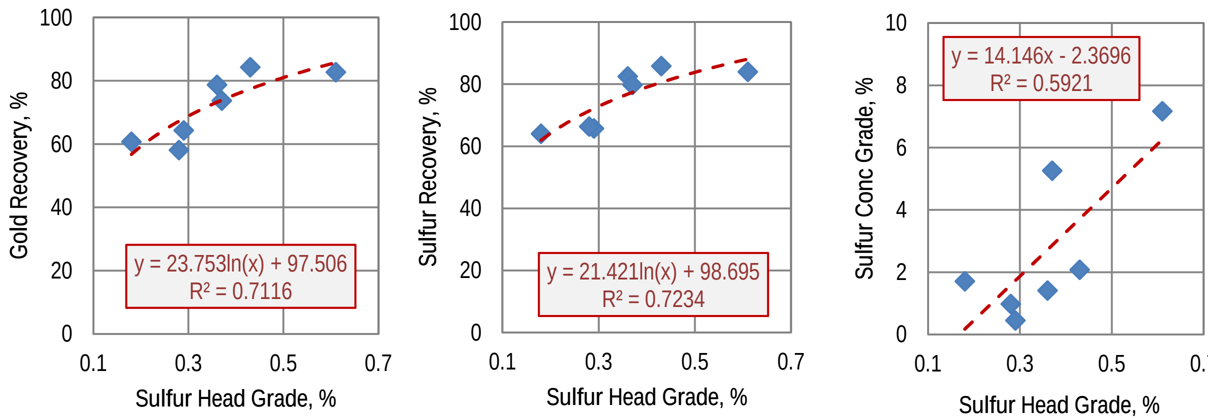


It is assumed, for the sake of the PFS projections, that leach recovery of gold in whole ore will improve by the same average (1.7%) as in the flotation tailings with the finer grind and this has been incorporated into the above analysis.

13.9.2.4 Historic Tailings Reprocessing

As the mine plan does not attempt to characterize the Historic Tailings material to be mined on a year-by-year basis, the metallurgical forecast for processing the Historic Tailings would also include a single performance parameter for the duration of the mine life. Based on tests on seven Historic Tailings samples, both gold and sulfur recovery are closely related to the sulfur head grade. Similarly the concentrate sulfur grade is linked to head grade. Using these correlations and a mine plan head grade of 0.33% sulfur, forecasts of gold recovery (71.2%), sulfur recovery (74.9%) and sulfur concentrate grade (2.3%) have been made.

Figure 13.18: Historic Tailings Sulfur Head Grade *vs* Process Parameters



Testwork on a blend containing 15% Historic Tailings and 85% Yellow Pine Early Production material showed there to be no adverse effects from blending these materials, such that the Project metallurgy of the blend would reflect the weighted mean metallurgical response of the two components.

13.9.3 Silver

The silver metallurgical forecasting equations have been summarized in Table 13.25.

Table 13.25: Silver Recovery Predictions

Deposit	Antimony Concentrate	Gold Concentrate	POX-CIL(2)	Tailings Leach	Whole Ore Leach
Yellow Pine Hi Sb	43%	30%	4%	10%	n/a
Yellow Pine Low Sb	n/a	73%	4%	10%	n/a
Hangar Flats High Sb	50%	30%	5%	6%	n/a
Hangar Flats Low Sb	n/a	80%	5%	6%	n/a
West End Sulfide	n/a	(1)	4%	59%	n/a
West End Mixed	n/a	(1)	4%	59%	n/a
West End Oxide	n/a	n/a	n/a	n/a	52%
Historic Tailings	n/a	73%	4%	10%	n/a

<u>*Note:*</u>
(1) West End Ag recovery: 0.912 x gold recovery - 1.05
(2) Percent of Ag reporting to POX that is recovered.

13.9.4 POX Leach Loss

There has been limited testwork completed defining the gold loss in oxidation and leaching of the flotation concentrates produced. It is recommended to apply a factor of 0.978 for Yellow Pine, 0.970 for Hangar Flats and 0.982 for West End to account for the gold remaining in the oxidized solids after leach.

13.9.5 Soluble Gold Loss

All barren solutions from the carbon elution and electrowinning circuits are considered internal recycling streams and should not contribute to gold losses. However, slurry streams which have passed through cyanide detoxification and are sent to the tailings facility will contain small amounts of soluble gold in solution unrecovered in the potential oxidized concentrate CIP and whole ore/flotation tailings carbon-in-leach (CIL) adsorption circuits and represent the only source of solution gold loss.

This soluble gold loss should be calculated from the steady state tailings pond solution volume plus pore solution in the settled solids and should not include reclaimed process water from the tailings facility which is also recycled. The calculation is performed with data sourced over a set time period and gives the cumulative mass of soluble gold loss, which as a fraction of the cumulative mass of gold fed to the leach gives the percent soluble gold loss over the set time period.

Limited test data indicates that the CIP adsorption efficiency averages approximately 94.9% and CIL adsorption efficiencies are greater than 98.5%. Further testwork is planned on the carbon and detoxification circuits in the feasibility testwork program. For the purposes of projecting PFS level metallurgical recoveries, losses due to carbon adsorption inefficiency in this program are only estimates. Soluble gold losses estimated against industry benchmarks are projected for this case at less than 1%, and possibly down to 0.5 - 0.2%, by mass, relative to gold in POX residues and oxide solids fed to leach (Ford, 2014).

For projection purposes, it is recommended to use a 0.8% loss in the calculations for all deposits (a factor of 0.992).

13.9.6 Metallurgical Summary

For convenience, all metallurgical recovery predictions for gold, silver, and antimony that have been developed in the preceding Sections, and that will form the basis of the PFS metallurgical recovery predictions, are provided in Table 13.26. The equations include all losses such that payable metal factors can be applied to these metallurgical recoveries. The only exceptions are the sulfur metallurgical regressions as these do not affect actual antimony, gold and silver metallurgy – just the mass balance of the circuit itself.

Table 13.26: PFS Metallurgical Recovery Prediction Equations Through POX and Leach

Deposit	Ore Type	Circuit	Metal	Metallurgical Recovery Equation (%)
West End	Oxide	Oxide	Au	Leach extraction = (85.151 x AuCN / AuFA + 9.8540) x 0.992
			Ag	Leach extraction = (Feed Ag Grade x 0.52) x 0.992
	Sulfide/ Mixed	Gold	Au	Cleaner flotation recovery = (-49.47 x AuCN / AuFA + 88.184) x 0.982 x 0.992
				Cleaner tailings leach extraction = (56.88 x AuCN / AuFA - 0.0813) x 0.992
			Ag	Cleaner flotation recovery = (0.912 x Cleaner Au Recovery - 1.05) x 0.04 x 0.992
				Cleaner tailings leach extraction = (100 – Cleaner Flotation Recovery) x 0.59 x 0.992
Yellow Pine	High Antimony Sulfide	Antimony	Sb	Antimony Cleaner Flotation Recovery = (10.353 x ln(Feed Sb Grade) + 93.341) x 0.981
			Ag	Antimony Cleaner Flotation Recovery = 43%
			Au	Antimony Cleaner Flotation Gold Loss = (3.1878 x ln(Feed Sb Grade) + 13.931) x 0.237
		Gold	Au	Rougher Flotation Gold Recovery = ((2.5239 x Feed Au Grade + 86.243) / 100 x (100 - Sb Cleaner Flotation Au Loss)) x 0.978 x 0.992
				Flotation Tailings Leach Extraction = (Au in Tailings x 0.10) x 0.992
			Ag	Rougher Flotation Silver Recovery = (100- Antimony Cleaner Ag) x 0.30 x 0.04 x 0.992
				Flotation Tailings Leach Extraction = [100 – Antimony Cleaner Ag – ([100 – Antimony Cleaner Ag] x 0.30)] x 0.10 x 0.992
	Low Antimony Sulfide	Gold	Au	Rougher Flotation Gold Recovery = (2.5239 x Feed Au Grade + 86.243) x 0.978 x 0.992
				Flotation Tailings Leach Extraction = (Au in Tailings x 0.10) x 0.992
			Ag	Rougher Flotation Silver Recovery = 2.9% x 0.992
				Flotation Tailings Leach Extraction = 2.7% x 0.992
Hangar Flats	High Antimony Sulfide	Antimony	Sb	Antimony Cleaner Flotation Recovery = (89.779 x (Feed Sb Grade ^ 0.0832)) x 0.967
			Ag	Antimony Cleaner Flotation Recovery = 50%
			Au	Antimony Cleaner Flotation Gold Loss = (3.4825 x (Feed Au Grade) ^ 1.1282) x 0.239
		Gold	Au	Rougher Flotation Gold Recovery = ((-8.0108 x Feed S Grade + 97.773) / 100 x (100 - Sb Cleaner Flotation Au Loss)) x 0.970 x 0.992
				Flotation Tailings Leach Extraction = ((-11.336 x Feed S Grade + 32.132)/100 x Au in Tailings) x 0.992
			Ag	Rougher Flotation Silver Recovery = (100 – Antimony Cleaner Ag) x 0.30 x 0.05 x 0.992
				Flotation Tailings Leach Extraction = [100 – Antimony Cleaner Ag – ([100 – Antimony Cleaner Ag] x 0.30)] x 0.06 x 0.992
	Low Antimony Sulfide	Gold	Au	Rougher Flotation Gold Recovery = (-8.0108 x Feed S Grade + 97.773) x 0.970 x 0.992
				Flotation Tailings Leach Extraction = ((-11.336 x Feed S Grade + 32.132) x Au in Tailings) x 0.992
			Ag	Rougher Flotation Silver Recovery = 80% x 0.08 x 0.992
				Flotation Tailings Leach Extraction = 4% x 0.992
Historic Tailings	Low Antimony Sulfide	Gold	Au	Rougher Flotation Gold Recovery = (23.753 x ln(Feed S Grade) + 97.506) x 0.978 x 0.992
				Flotation Tailings Leach Extraction = (Au in Tailings x 0.20) x 0.992
			Ag	Rougher Flotation Silver Recovery = 2.9% x 0.992
				Flotation Tailings Leach Extraction = 2.7% x 0.992

Notes: *AuCN / AuFA ratio is expressed as a decimal not as a percent; Feed Au Grade and Feed Ag Grade are expressed in g/t; Antimony Cleaner Ag, Cleaner Au Recovery, Feed Sb Grade, Feed S Grade, Au in Tailings and Sb Cleaner Flotation Au Loss are expressed in %.*

13.10 METALLURGICAL OPPORTUNITIES

13.10.1 Tungsten Recovery

While a tungsten resource remains undefined, portions of the Yellow Pine, Hangar Flats and West End deposits contain potentially recoverable tungsten in the form of scheelite (Blake, 2012). Historically, from 1940 through 1945, tungsten was recovered into two concentrates from Yellow Pine ore by flotation (high-grade concentrate) and tabling (low-grade concentrate) after the completion of sulfide flotation. High-grade materials were placer-mined which included ultra-violet (UV) hand sorting of trommel oversize and jigging of undersize. Cleaning of the various jig middlings included magnetic separation and tabling. A tungsten flotation recovery circuit was also started up to treat the sulfide flotation tailings (Mitchell, 2000).

A scoping test program investigating the recovery of tungsten was conducted on two small variability composites, a high-grade sample from Hangar Flats assaying 1.6% tungsten, and a lower grade sample from Yellow Pine assaying 0.42% tungsten. The limited program included a gravity release study, centrifugal gravity separation, tabling and flotation scoping testwork, in each case of gold-bearing sulfide flotation tailings (Gajo, 2014b).

Sulfide flotation of the Hangar Flats and Yellow Pine samples left 76.3% and 65.5% of the tungsten in the tailings. In both cases close to 80% of this tungsten was in the minus 53 µm fraction. Due to its fine size, gravity recovery work was not successful. The most effective rougher recovery method in both cases was scheelite flotation, with 95% and 76% recovery from the Hangar Flats and Yellow Pine samples. Mass pull rates were high making this no more than a pre-concentration step, and the limited testing of further concentration (cleaning) by gravity on flotation concentrates was not successful.

13.10.2 Finer Primary Grind

It was noted in the Master Composites summary sections that a finer grind was found to be beneficial to gold recovery for the West End Sulfide composite, which according to the PFS potential mining schedule represents approximately 29% of the mill feed to the plant over the life of the mine. For the WES composite, it was noted that the reduction in grind size from a P_{80} of 84 µm to 56 µm improved gold recovery in rougher flotation by about 4%. A comparison between the extra power and/or equipment required to achieve this finer grind and the increased amount of gold recovered would need to be studied (Gajo, 2014a).

It should be noted that a finer grind size did not appear to be any benefit to the Yellow Pine low antimony ores nor the Hangar Flats low antimony ores, although the data from the Hangar Flats low antimony tests is weak and further testwork is recommended. The effect of finer grind on high antimony ores was also not tested and should be evaluated to gauge the effect of a finer grind on possible sliming of the antimony flotation circuit.

13.10.3 Elimination of POX Countercurrent Decantation Circuit

Cyanide leaching Yellow Pine POX slurries was completed at SGS Lakefield and included a study into the effect of degree of slurry washing on cyanidation and gold recovery (Jackman, 2014a). Samples from three of the pressure oxidation runs underwent five cyanidation leaches to determine the extraction of gold. There were no regrinding tests performed. Three of the tests were straight bottle roll cyanidation leaches after 100% of hot cure solution was washed out of the solids, while two of them evaluated the effect of the degree of hot cure slurry washing on leach recovery.

Tests evaluating the effect of degree of slurry washing on gold extraction and reagent consumptions showed that there was less than 1% decrease in gold extraction when moving from fully washed slurry through to unwashed (Table 13.27). However, throughout that change, the cyanide consumption increased 7-fold and lime consumption

29-fold. The balance of capital and operating costs for washing thickeners versus reagent consumption with less washing is recommended to be studied further in the future.

Table 13.27: Summary of Slurry Washing Effects on Cyanidation

POX Discharge Slurry Washing Conditions	Reagent Consumption		Extraction
	NaCN (kg/t)	CaO (kg/t)	Au (%)
100% Water	0.34	3.9	97.9
34% POX PLS / 66% DI Water	1.29	59.4	97.8
100% POX PLS	2.45	114.0	97.2

13.11 ALTERNATIVE PROCESSES

13.11.1 Antimony Concentrate Processing

Two scoping studies were undertaken to evaluate the options for antimony concentrate processing by Midas Gold, as opposed to direct sales of concentrate to a separate party. The two options evaluated were roasting (at Kingston Process Metallurgy, Kingston, Ontario) and leach-electrowinning (at SGS Lakefield, Lakefield, Ontario).

The concentrates sent for the two studies were produced from a high-grade antimony mixture of material from the Hangar Flats and Scout Ridge prospect areas of the Project. These were produced from 26 x 10 kg batch tests with two stages of cleaning of the antimony concentrates which produced approximately 11 kg of concentrate at an average grade of 50.4% antimony.

13.11.1.1 Stibnite Roasting

Roasting scoping studies were conducted at Kingston Process Metallurgy in a two phase program involving static kiln tests at three temperatures (700, 800, and 900 ºC) followed by two rotary kiln runs at temperatures near the optimum identified in the static tests. Results of the tests were forwarded to SGS for cyanidation of the calcines to evaluate amenability to gold extraction. Based on the results of the rotary kiln tests, a preliminary heat and mass balance was also evaluated (Pettingill, Davis, & Roy, 2013).

Results of the static kiln tests showed the best antimony removal at temperatures of 800 ºC and higher, with greater than 99% removal of antimony from the concentrates. Precipitates from the condensation zone ranged from 79.3 - 83.6% Sb, 0.77 - 0.81% As and 0.08 - 0.24% Fe. The final rotary kiln results showed that at 950 ºC, 99.9% of the antimony and 95% of the sulfur off-gassed (as SO_2) in the first 2 hours. Cyanidation of the calcines was able to extract 95% of the gold remaining in them.

13.11.1.2 Stibnite Leach – Antimony Electrowinning

The second study conducted on the concentrates was done at SGS Lakefield and involved scoping testwork into a stibnite leach – antimony electrowinning process. A significant potential upside to the leach-electrowinning program is that the leach residues from the process would be available for reprocessing in the autoclave, rendering that gold recoverable.

Scoping testwork involved investigating three leach methods: ferric chloride, caustic, caustic sulfur and caustic sulfide, followed by Hull cell electrowinning. Leach parameters investigated included reagent concentration and leach temperature; leach tests were conducted with kinetic samples pulled to assess the extraction vs. time curve for each. The final solutions were placed into a Hull electrowinning cell to test deposition of antimony on the cathode, configuration of the Hull cell tested current densities from 0 - 500 ampere per square meter (A/m^2). Parameters investigated in electrowinning included: temperature, degree of mixing, current intensity, current density and cathode type: stainless steel, copper or brass (Lupu & Gladkovas, 2014).

Caustic leach: Results of the caustic leach showed that antimony extraction of 99.5% could be achieved with a 10% NaOH solution at 25ºC in 3 hours, when conducted at 2% solids in the leach. All tests exceeded 90% extraction of antimony, achieved within one hour. Gold was not leached and silver dissolution was less than 10%. In the single test where it was measured, 94% of the arsenic was extracted.

Caustic sulfide leach: Antimony extraction of 99.9% was achieved in the first few hours of the leach with both sulfur sources (sodium sulfide or elemental sulfur) under all test conditions. Use of sulfur as the sulfur source appeared to leach about 24% of the gold from the concentrate while silver extraction was about 10%. Use of sodium sulfide as the sulfur source leached less than 10% of the gold and leached 26 - 30% of the silver. Greater than 75% of the arsenic also appears to be leached in the tests.

Ferric Chloride: Results of the ferric chloride leach showed that at 90 ºC and 150 g/L sodium chloride, greater than 93% extraction of antimony could be achieved. The parameters tested resulted in extractions ranging from 55 to 93%. There was no indication of gold leaching in any of the tests, while silver extraction ranged from about 30 to 67%. Arsenic extraction was varied as well, with tests leaching from 25 to 85% of the arsenic.

Electrowinning of antimony from all solutions was successful, though the degree of metal adhesion varied with each leach solution and cathode material.

The caustic sulfide leach was tested in a brief locked cycle test employing leaching, electrowinning and re-leaching to provide preliminary insight into the suitability of the spent solutions from electrowinning for re-leaching a new batch (Gladkovas, 2014).

In both leach cycles, antimony extraction was close to 99%. Current efficiency dropped quickly in electrowinning due to depletion of the antimony in solution, but by the end of the second leach, antimony loading had risen to the point where significantly more efficient electrowinning could be expected. Initial indications are, therefore, that the process will prove to be workable in commercial operation – however no analyses of the electrowon product were obtained to explore its potential marketability. Mineralogical analyses of the leach residues from the study indicated that the gold-bearing pyrites and arsenopyrites were intact and likely to be available for processing in the autoclave with other gold concentrates. The state of any remaining silver was not investigated and should be evaluated in the future.

13.11.1.3 Neutral pH Pressure Cyanidation of Antimony Concentrates

Conventional cyanidation of otherwise free-milling gold is not possible in antimony-rich materials as the antimony consumes large amounts of the cyanide at high pH levels. Accordingly, neutral pH cyanidation is practiced under pressure using a pipe reactor at Consolidated Murchison in South Africa. Such a process may allow for extraction of silver and some of the gold from the antimony concentrates, and should be tested in due course. Mild pre-oxidation of the stibnite has also been proposed as an alternative, whereby the stibnite surface is sufficiently oxidized to be passivated from reaction with the cyanide.

13.11.2 Gold Concentrate Processing

The refractory nature of the gold-bearing minerals in Midas Gold ores necessitates the oxidation of the sulfides in order to make the gold amenable to cyanide leaching. Various options are available commercially and each was researched as a possible method for Midas Gold at one point or another. A brief summary of each of the processes is provided below, along with any relevant testwork that has been completed.

13.11.2.1 Roasting

Historic roasting of antimony and refractory gold concentrates in the 1950s at the Yellow Pine smelter is described by Huttl (1952, *via* Mitchell, 2000) and summarized below. Assays of the concentrates sent to roasting are summarized in Table 13.28 (converted to metric units).

Table 13.28: Assays of Historic Concentrates Sent to Roaster

Sample	Head Grades				
	Au (g/t)	Ag (g/t)	Sb (%)	As (%)	S (%)
Gold Concentrate	85.7	102.9	4.0	9.0	35.0
Antimony Concentrate	20.6	582.9	46.0	1.8	22.0

The site had an eight hearth roaster with a temperature range of 370ºC - 730ºC which only needed a burner on one hearth due to the high sulfur content and "self-roasting" nature of the concentrates. Calcines were cooled on a rotary cooling conveyor and stored in bins; the calcines were reported to assay 1.5% S, 0.5% As and 3.0% Sb and fed the on-site smelter for doré production.

No roasting studies were conducted during the PEA or PFS program, but testwork has been completed on concentrates from Yellow Pine and West End in the past. In 1978, roasting of West End sulfide concentrates was conducted for Superior by Britton Research Ltd. The report indicated that roasting with an excess of lime at 700 - 750ºC fumed off 2.2 - 4.5% of the sulfur and 14.1 - 17.2% of the arsenic, cyanidation of the calcines was able to extract 75.2 - 76.6% of the gold in 72 hours. Reagents consumed ranged 18.5 - 21.0 kg/t for cyanide and no lime in the cyanidation tests. It was noted that roasting was not complete in either case and that more complete roasting should lead to "a marked reduction in the cyanide consumption" (Britton, 1978).

Roasting of Yellow Pine concentrates was conducted in 1987 for Hecla by Lakefield Research. The testwork was conducted at 570 - 670ºC with equal mass of nepheline syenite. There was greater than 95% oxidation of sulfides and elimination of 77 - 87% of arsenic and 86 - 96% of antimony in roasting. An acid leach of the calcines was conducted followed by cyanidation, which extracted between 62 - 74% of the gold from the calcines. Reagents consumed were 13.1 - 25.8 kg/t for cyanide and 8.9 - 13.0 kg/t for lime in the cyanidation tests. The best gold extraction was achieved at the 570ºC roasting temperature (Rollwagen, 1987).

13.11.2.2 Albion

The Albion process has not been tested on any concentrates to date. While the energy requirements for ultra-fine grinding are typically high, the site has access to relatively cheap hydroelectric power which, when combined with the lower reagent costs, may make this a feasible alternative. However, Albion was not envisioned to result in appreciable capital or operating cost savings, it was not envisioned to improve metallurgical recoveries, and it was not felt to be a technology that was sufficiently proven in a commercial setting to pursue at this time.

13.11.2.3 Biological Oxidation

Biological oxidation of the refractory sulfides prior to cyanidation for gold recovery has been tested in the West End and Yellow Pine sulfide concentrates and ores since the late 1980s and most recently on a PEA scale for the current Project.

Past Testwork

Highly refractory gold-bearing sulfide concentrates from West End, tested by Coastech Research Inc. in 1987 (Broughton, 1987) yielded 83.6% and 78% extraction of gold and silver respectively, after 85% of the sulfides were oxidized.

Yellow Pine concentrate and ore were tested by Giant Bay Biotech Inc. in 1987 (Hackl, 1987). With 93% of the sulfides oxidized, gold extraction by cyanidation improved from 10.3% to 97.2%. A continuous bio-oxidation run was then operated, the best conditions proving to be three stages of bio-oxidation over 5.2 days at 32% solids by weight and recycle of as much bio-leachate as was practical. Under those conditions they were able to achieve 91% gold recovery with cyanide consumptions in the 1.9 - 2.6 kg/t range. They recommended either a longer retention time or finer grind to further improve gold extraction.

Yellow Pine ore was tested in a heap bioleach for Hecla by the University of Idaho in 1992 (Harrington, Bartlett, & Prisbrey, 1993). The tests were completed in columns representing seven different size fractions, each utilizing cultured host rock bacterial colonies. Cyanidation was then performed on the oxidized material after size reduction to nominal minus 53 µm. Up to 72% of the gold was leachable after 360 days of bio-oxidation in the heap. Models predicted bio-oxidation would be required for two years to achieve over 90% extraction.

Current Testwork

Portions of the PEA sulfide concentrates from Yellow Pine, Hangar Flats and West End that underwent the pressure oxidation testwork were also tested through a bio-oxidation study at SGS Lakefield (Jackman, 2014b) for evaluation of applicability of the BiOx® process.

Data gathered from the study was provided to Biomin South Africa (Pty) Ltd (Biomin) for final evaluation and interpretation along with an order of magnitude capital and operating expense study (Olivier, 2014). The tests were conducted at 13% solids, between 38 and 42°C and a pH range of 1.4 - 1.6 controlled with acid and limestone additions. Cyanidation was conducted on rinsed residues as a CIP bottle roll at 35% solids over 48 hours at a maintained NaCN concentration of 1 g/L.

Acidulation of the concentrates was performed at the start of each test to ensure that the bacterial inoculum remained viable. This required an average of 78 kg/t sulfuric acid for Yellow Pine and Hangar Flats concentrates and 225 kg/t sulfuric acid for the high carbonate West End concentrates. Once started, the tests were acid producing and to maintain pH each consumed a maximum of 182 kg/t of limestone over the thirty-day runs and averaged 153 kg/t at twenty days and 93 kg/t at ten days.

Levels of microbial activity and oxidation of sulfides were monitored through electromotive force (EMF) potential readings on a silver/silver chloride electrode, in millivolts (mV), throughout the study. Readings of over 550 mV indicated increased microbial activity, while readings over 700 mV indicated active oxidation of sulfides by the bacteria. Results of bio-oxidation for all three concentrates were similar (see Table 13.29), with greater than 90% oxidation of sulfide achieved in the first ten days of batch testing and greater than 95% by the twentieth day (15^{th} in the case of West End concentrate). Extraction of gold was highest with West End, achieving 95.5% extraction after ten days of bio-oxidation and 97.8% after thirty days; silver extractions ranged from 75 - 79%. Next highest was Yellow Pine, achieving 92.1% gold extraction after ten days of bio-oxidation and 96.9% after thirty days; silver extractions ranged from 73 - 93%. Hangar Flats had the lowest gold extractions, achieving 86.9% gold extraction after ten days of bio-oxidation and 94.2% after thirty days; silver extractions ranged from 79 - 86%.

Table 13.29: Summary of Batch BiOx® Testwork Results

Test No.	Time (days)		$S^=$ Oxidation (%)	Recovery in CN Solution + Carbon	
	EMF >550 mV	EMF >700 mV		Au	Ag
Hangar Flats Concentrate					
Bio 1R	4	1	50.1	55.4	58.8
Bio 2	10	7	91.8	86.9	79.1
Bio 3	14	10	92.7	92.4	83.8
Bio 4	20	18	97.1	92.4	86.3
Bio 5	25	21	97.4	93.1	83.0
Bio 6	30	28	97.9	94.2	86.2
West End Concentrate					
Bio 7	4	0	26.9	56.8	42.4
Bio 8	8	6	93.3	95.5	80.8
Bio 9	14	11	98.8	97.5	79.4
Bio 10	19	16	98.4	96.4	74.7
Bio 11	24	21	99.0	97.8	77.2
Bio 12	29	28	99.0	97.8	77.1
Yellow Pine Concentrate					
Bio 13R	3	1	53.1	63.1	75.8
Bio 14	9	6	91.6	92.1	92.9
Bio 15	13	10	93.9	94.6	91.5
Bio 16	20	18	99.0	97.2	79.4
Bio 17	25	23	99.3	95.9	73.6
Bio 18	30	28	99.2	96.9	76.2

In leaching, the maximum cyanide consumption in the tests was an average of 7 kg/t, with the 20 day tests averaging 4.5 kg/t and the 10 day tests averaging 5.6 kg/t. The maximum lime consumption was an average of 40.6 kg/t, with the 20-day tests averaging 33.3 kg/t and the 10-day tests averaging18.1 kg/t.

Biomin's analyses of the data resulted in a recommendation that a BiOx® facility would require a 5-day residence time at 25% solids in the feed for the bioreactors to achieve 95% sulfide oxidation. The design assumes an average 13.8% sulfide sulfur in the feed, operation at 40 °C, nutrient feed at 3.9 kg/t of concentrate and use 67 kg/t of 75% pure limestone to maintain pH. A small amount of sulfuric acid would also be required at commissioning. The reactors would require external cooling to maintain temperature and blowers to provide air to the reactors.

Overall the residence time, oxidation and cyanidation findings were similar to the Hecla/Giant Bay study from 1987.

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14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

The Mineral Resource Statement presented herein represents the third mineral resource evaluation prepared for Midas Gold by qualified independent consultants for the Project in accordance with the Canadian Securities Administrators' National Instrument 43-101 and includes the maiden resource estimate for the historic tailings deposit.

This section describes the mineral resource estimation methodology and summarizes the key assumptions used. In the opinion of Garth Kirkham, P.Geo., Qualified Person, the mineral resource estimates reported herein are a reasonable representation of the mineral resources found within the Project at the current level of sampling. The mineral resources were estimated in conformity with generally accepted Canadian Institute of Mining and Metallurgy (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" and are reported in accordance with the Canadian Securities Administrators' NI 43-101. It is important to note that mineral resources that are not mineral reserves do not have demonstrated economic viability. Mineral resource estimates do not account for mine-ability, selectivity, mining loss and dilution. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated.

The mineral resource evaluation reported herein is current and supersedes earlier mineral resource estimates completed for Midas Gold including:

- Technical Report on Mineral Resources for the Golden Meadows Project, Valley County, Idaho, dated June 6, 2011 (SRK, 2011).
- Preliminary Economic Assessment Technical Report for the Golden Meadows Project Idaho, September 21st, 2012 (SRK, 2012).

The mineral resource estimates were reviewed and verified by Garth Kirkham, P.Geo., the Independent Qualified Person for the mineral resource estimates for the Project and included in this Report. Midas Gold's field work on the Project from 2009-2014, including drilling, was carried out under the supervision of Chris Dail, CPG and Richard Moses, CPG, who were Midas Gold's senior geologists responsible for certain aspects of the programs during the periods they were employed by Midas Gold.

The general mineral resource estimation methodology for all deposits involved the following procedures:

- review of the geologic model and structural controls on mineralization;
- database verification;
- validation and verification of historic databases;
- data exploration, compositing and capping;
- construction of estimation domains for gold, antimony and silver;
- spatial statistics;
- block modeling and grade interpolation;
- mineral resource classification and validation;
- assessment of "reasonable prospects for economic extraction;" and

- preparation of the mineral resource statement.

Detailed mineral resource evaluation methodologies are discussed in subsequent sections for Hangar Flats (Section 14.2), West End (Section 14.3), Yellow Pine (Section 14.4), and Historic Tailings (Section 14.5). An assessment of reasonable prospects for eventual economic extraction and mineral resource statements are presented in Sections 14.6 and 14.7, respectively.

14.2 HANGAR FLATS

14.2.1 Mineral Resource Estimation Procedures

The mineral resource estimate for Hangar Flats is based on the validated and verified drill hole database, interpreted geologic units and fault structures, digitized underground historic workings, and light detection and ranging (LiDAR) topographic data. The geologic modeling and estimation of mineral resources was completed using the commercial three-dimensional block modelling and mine planning software packages Geovia GEMS™ 6.6 and Micromine™ version 14; geostatistical analysis was completed using Isaaks & Co.'s SAGE2001™ software package.

14.2.2 Drill Hole Database

The drill hole database was supplied by Midas Gold as an Excel Workbook that contained collar locations surveyed in UTM grid coordinates, drill hole orientations with downhole surveys, assay intervals with gold, antimony, and silver, and geologic intervals with rock types. The database provided for the mineral resource estimate contained data for 256 separate drill holes representing both historic and modern drilling programs, as previously described in Section 10. The drill holes were reviewed and certain drill holes were not considered for use in mineral resource estimation, including air-track, rotary and pre-collar drill holes. Some pre-1953 drill holes were used to guide construction of estimation domains but were not used in the mineral resource estimates, including four Bradley Mining Company (Bradley) MC-series holes with sample lengths >6 m, all 1929 BMC-series holes with incomplete supporting documentation, and six DMA-series holes for which grades were not corroborated by Midas Gold confirmation drilling. After removal of select drill holes and non-bedrock assay intervals, the final database used for mineral resource estimation contained 219 drill holes with gold assays, of which 169 also have antimony assays and 144 have silver assays (as shown in Table 14.1). The vast majority of assay lengths are 1.52 m (5 ft) with sample intervals as long as 9.1 m with an overall average of 1.6 m. Modern era drill holes (post-1953) were typically drilled at a spacing of 30 to 50 m.

Table 14.1: Drill Hole Information used in the Hangar Flats Mineral Resource Estimate

Company	Gold			Silver			Antimony		
	# Holes	# Samples	Meters	# Holes	# Samples	Meters	# Holes	# Samples	Meters
Bradley	38	1,629	2,900	0	0	0	33	1,412	2,504
El Paso	3	112	364	2	89	274	2	46	128
Hecla	22	715	1,106	8	185	299	0	0	0
Midas Gold	134	18,323	31,386	134	18,323	31,386	134	18,291	31,328
USBM	22	807	1,225	0	0	0	0	0	0
All	219	21,586	36,981	144	18,597	31,959	169	19,749	33,959
Note: Drill hole information includes un-sampled intervals.									

14.2.3 Geologic Modeling

Mineralization in the Hangar Flats deposit occurs in intrusive rocks associated with the Atlanta Lobe of the Idaho Batholith consisting of quartz monzonite and alaskite compositions. Mineralization is localized along an overall north to south striking fault zone and also along northeast striking splay faults and dilatational fault jogs. The interaction of these structural sets, one steeply dipping and one shallowly dipping, provided the ground preparation favorable for deposition of gold and antimony. Post-mineral dikes intrude the Idaho Batholith and consist of rhyolite and diabase.

The geologic model for Hangar Flats is based on a generalized single rock type model of quartz monzonite with solids representing the post-mineralization intrusive dikes and historic underground workings. Modeled structures include the Meadow Creek Fault Zone (MCFZ) solid (representing zones of breccia, gouge, and cataclasite), and subsidiary northeast striking splay faults. Midas Gold provided a topographic surface derived from 1 m gridded LiDAR flown in 2009, and a surface representing the current top of bedrock based on drill hole data. Historic underground workings of the Meadow Creek Mine (consisting of levels, raises, shafts, adits, and stopes) were modeled from geo-rectified historic maps in both plan views and section views.

Figure 14.1 depicts a plan view of the Hangar Flats geologic model. The historic underground workings were also modeled and the resulting solids were used to remove mined-out volumes from the reported mineral resource.

The MCFZ is the principal structure controlling mineralization. The MCFZ varies in width from 40 to 100 m and varies in dip from 80 degrees west to 45 degrees east. Gold mineralization and antimony mineralization form a corridor around the eastern boundary of the MCFZ at the intersections of the MCFZ and numerous low angle faults. The geometry and spatial extents of mineralization on the west side of the MCFZ is uncertain due to very low density of drilling. The primary occurrence of gold is similar to Yellow Pine, within quartz-sulfide veining, irregular masses within breccia and disseminated in the country rock as sulfides replacing biotites. As discussed in Section 7, antimony mineralization occurred later than gold but utilized many of the same structures.

14.2.4 Estimation Domain Modeling

The Hangar Flats gold and silver estimates utilize a grade shell and three estimation domains to define regions with different structural controls on mineralization. The antimony estimate utilizes an antimony shell. Solids representing dikes and historic underground workings were used for density assignment and data filtering only.

A gold grade shell was constructed based on an indicator Kriging estimate at a 0.25 g/t gold cutoff grade. Silver also utilizes the gold shell as silver correlates strongly with gold. A 0.1% antimony shell was manually constructed based on the underground workings, assay composites, and the geologic model. While some minor oxidation is observed within Hangar Flats, all material is assumed to be sulfide for the purpose of the mineral resource estimation.

Three estimation domains were generated for gold based on geologic interpretations, structural interpretations and the 0.25 g/t gold shell (Figure 14.2). Domain 1 is generally controlled by the MCFZ and contains steeply dipping gold mineralization. Domain 2 is bounded by Domain 1 on the west side and subdivides moderately dipping mineralization from mineralization in the fault corridor. The final gold domain contains all other blocks estimated in the model but not within the gold shell. Silver was also estimated within the gold domains. The antimony shell occurs entirely within the gold shell and contains nearly all of the potentially economic antimony mineralization. Descriptive statistics for raw assay data within the final estimation domains are shown in Table 14.2.

Variations in the mean grade of gold, antimony and silver across the domain and shell boundaries were examined using contact plots to determine domain boundary treatment during estimation. As a result, the boundary between Domains 1 and 2 is treated as a soft boundary and the antimony shell is treated as a hard boundary during estimation.

Figure 14.1: Geologic Model for Hangar Flats

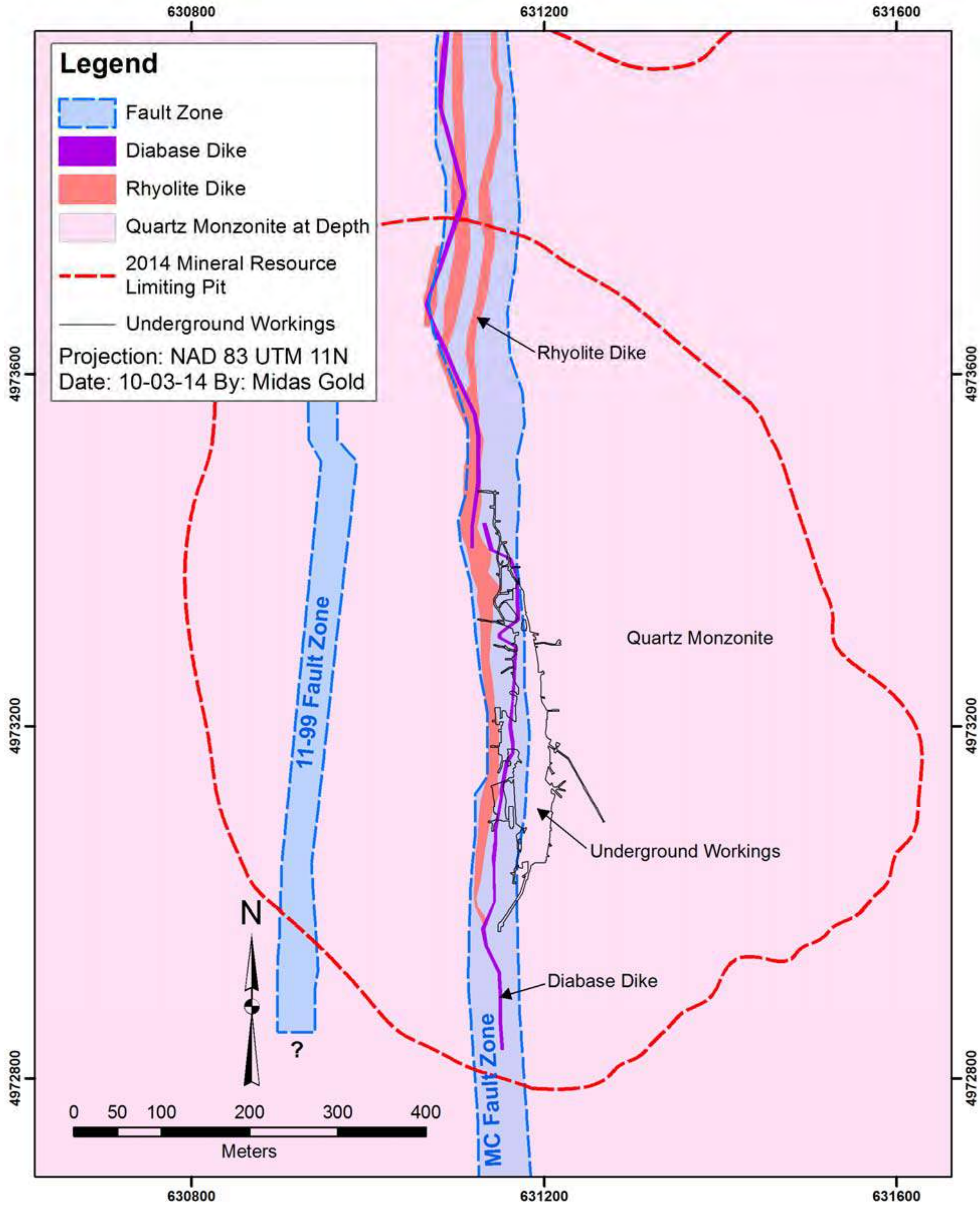


Figure 14.2: Estimation Domains for Hangar Flats

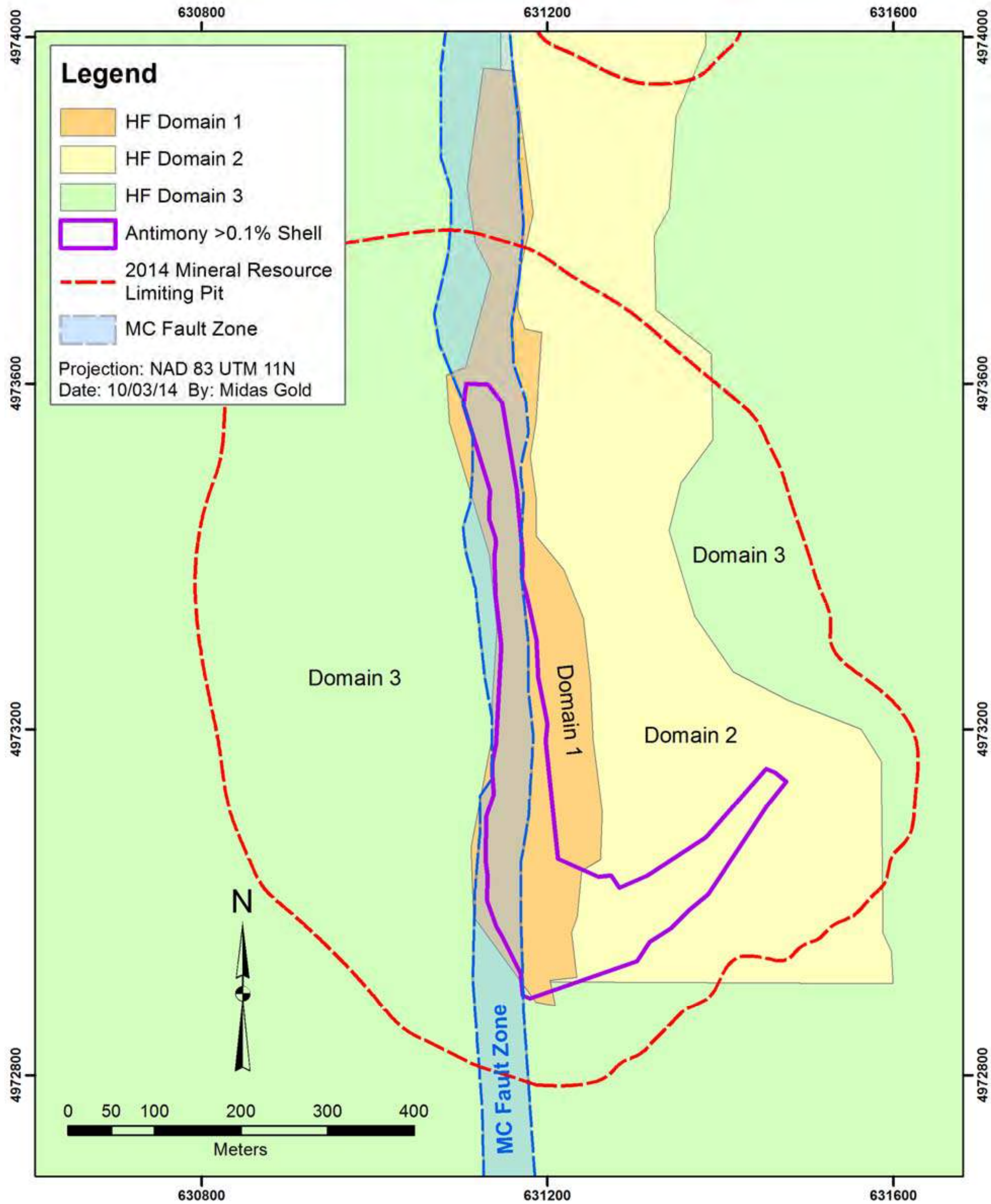


Table 14.2: Hangar Flats Raw Assay Descriptive Statistics by Estimation Domain

Statistic	Au (g/t) by Domain				Ag (g/t) by Domain				Sb (%) Relative to Shell		
	All	1	2	3	All	1	2	3	All	Sb Shell	Outside
Number	21,142	5,887	11,387	3,868	18,315	5,200	10,204	2,911	19,583	4,729	14,854
Mean	0.600	1.221	0.464	0.054	2.5	6.9	0.8	0.7	0.102	0.388	0.011
Standard Deviation	1.365	1.958	1.069	0.207	35.4	64.7	7.1	13.3	0.854	1.696	0.107
Minimum	0.003	0.003	0.003	0.003	0.0	0.3	0.0	0.3	0.000	0.000	0.000
Lower Quartile	0.005	0.034	0.008	0.003	0.3	0.3	0.3	0.3	0.001	0.005	0.001
Median	0.052	0.397	0.065	0.003	0.3	0.8	0.3	0.3	0.003	0.011	0.002
Upper Quartile	0.507	1.605	0.377	0.034	0.9	2.7	0.6	0.3	0.006	0.164	0.004
Maximum	24.8	24.8	17.85	5.143	3,160	3,160	679	709	35.00	35.00	8.290
Coefficient of Variation	2.276	1.603	2.305	3.834	14.0	9.4	8.6	20.1	8.395	4.376	10.009
95% Percentile	3.12	5.22	2.37	0.21	4.8	14	2.9	0.5	0.300	1.626	0.014
98% Percentile	5.12	7.376	3.909	0.549	10	55	4.3	1.2	0.947	4.200	0.082
99% Percentile	6.857	8.874	5.123	0.895	30.0	103.0	5.6	3.2	2.022	7.033	0.173
Note: Drill hole information excludes un-sampled intervals											

14.2.5 Compositing

Gold, antimony and silver were composited downhole on 3.048 m (10 ft) intervals across geologic and domain boundaries. Composites associated with specific pre-1953 drill holes (as previously discussed), those falling within the rhyolite solid and those <0.61 m (2 ft) in length were removed and are not utilized in the estimation nor were they included in the statistical analysis. Pre-1953 historical data retained for the mineral resource estimate comprise 11% of the total data set. Table 14.3 shows statistics for the raw, un-capped composites.

Table 14.3: Hangar Flats Raw Composite Statistics by Estimation Domain

Statistic	Au (g/t) by Domain				Ag (g/t) by Domain				Sb (%) Relative to Shell		
	All	1	2	3	All	1	2	3	All	Sb Shell	Outside
Mean	0.577	1.185	0.452	0.051	2.493	6.988	0.817	0.442	0.085	0.325	0.009
Standard Error	0.011	0.030	0.012	0.004	0.399	1.402	0.061	0.055	0.006	0.023	0.001
Median	0.091	0.504	0.107	0.003	0.250	0.920	0.250	0.250	0.003	0.019	0.002
Standard Deviation	1.171	1.677	0.893	0.175	39.172	73.242	4.406	2.287	0.576	1.133	0.053
Minimum	0.003	0.003	0.003	0.003	0.010	0.250	0.010	0.250	0.000	0.000	0.000
Maximum	14.74	14.74	13.12	2.58	3,160	3,160	236.7	70.03	25.54	25.54	1.65
Count	11,132	3,106	5,802	2,224	9,660	2,729	5,202	1,729	10,222	2,478	7,744
Coefficient of Variation	2.03	1.41	1.97	3.41	15.71	10.48	5.39	5.17	6.74	3.48	6.14

14.2.6 Evaluation of Outliers

To mitigate risk associated with use of high-grade statistical outliers, capping grades were selected for each estimation domain after declustering and weighting raw composite data. Capping grade was evaluated through log probability plots and through analysis of contained metal within deciles and centiles, following the Parrish Method (Parrish, 1997). Both methods yielded similar results and final composite capping levels are shown in Table 14.4. Descriptive statistics for capped composites are show in Table 14.5.

Table 14.4: Hangar Flats Composite Capping Grades by Estimation Domain

Statistic	Au (g/t) by Domain		Ag (g/t) by Domain		Sb (%)
	Domain 1	Domain 2	Domain 1	Domain 2	Sb Shell
Number	3,106	5,802	2,729	5,202	2,478
Maximum Value	14.7	13.12	3,160	236.66	25.54
Cap Value	10	10	225	11	8
Number Capped	9	1	8	14	10
Mean Uncapped	1.19	0.45	6.99	0.82	0.33
Mean Capped	1.18	0.45	4.91	0.71	0.31
Metal Removed	<0.5%		>30%		6%

Table 14.5: Hangar Flats Descriptive Statistics for Capped Composites

Statistic	Au (g/t) by Domain		Ag (g/t) by Domain		Sb (%)
	Domain 1	Domain 2	Domain 1	Domain 2	Sb Shell
Mean	1.181	0.452	0.723	0.641	0.307
Standard Error	0.030	0.012	0.048	0.018	0.018
Median	0.504	0.107	0.250	0.250	0.019
Standard Deviation	1.651	0.886	3.988	1.475	0.905
Minimum	0.0025	0.0025	0.01	0.01	0.0002
Maximum	10.00	10.00	236.66	70.03	8.00
Count	3,106	5,802	6,931	6,931	2,478
Coefficient of Variation	1.40	1.96	0.551	2.30	2.95

14.2.7 Statistical Analysis and Spatial Correlation

Correlogram models were developed using the SAGE2001™ software package to guide the search ellipses and establish spatial correlation and sample weighting for the estimates. The nugget effect was derived using down-hole correlograms. Correlograms were developed for gold within estimation Domains 1 and 2, for antimony within the antimony shell, and for silver within Domain 1 and Domains 2 + 3 combined, as silver occurs both east and west of the MCFZ. Gold and silver correlograms demonstrate spatial correlation along primary structural trends while antimony is somewhat oblique. Gold and antimony correlograms were verified using indicator variograms. The correlogram parameters are summarized in Table 14.6.

Table 14.6: Correlogram Models for Hangar Flats

Element	Estimation Domain	Ellipse Axes Azimuth/Plunge			Nugget C0	Sill C1 and C2	Modeled Ranges Model 1 / Model 2 (m)			Type
		1st	2nd	3rd			1st	2nd	3rd	
Au	1-MCFZ	300/78	355/-7	264/-10	0.248	0.393	36/75	33/538	20/48	Exp
						0.359				
	2-Splay Faults	321/70	350/-18	77/9	0.193	0.439	19/176	8/545	55/253	Exp
						0.367				
Sb	1- Sb Shell	295/13	26/2	125/77	0.365	0.454	11/42	11/256	38/68	Exp
						0.182				
Ag	1	348/-3	33/86	258/3	0.414	0.48	20/360	52/27	12/219	Exp
						0.128				
	2 & 3	157/27	333/65	66/1	0.274	0.726	149	65	77	Exp
Note: Negative plunge is downward										

14.2.8 Block Model Parameters and Grade Estimation

The Hangar Flats block model used 12.192 x 12.192 x 6.096 m (40 x 40 x 20 ft) blocks with coordinates defined in Table 14.7. The selected block size is approximately 25% of the median spacing of modern era drill holes and 35% of the median spacing of all drill holes and is consistent with conceptual mining bench heights. Blocks were discretized into a 4 x 4 x 2 array of points.

Table 14.7: Block Model Definition for Hangar Flats

Deposit	Dimension (m)			Origin (m)			Number of Blocks			Rotation
	X	Y	Z	X	Y	Z	X	Y	Z	
Hangar Flats	12.192	12.192	6.096	630,509.5	4,972,588	1,569.7	107	150	138	0
Note: Block centroid, NAD83 Zone 11N Datum										

The Hangar Flats database contains 988 density measurements for different rock types ranging from 2.01 g/cm^3 to 5.46 g/cm^3; the majority of the measurements were done onsite using the water displacement method, supported by independent third party estimates for verification. Density measurements were grouped using the geologic model wireframes and density values were calculated for each rock type represented in the geologic model after capping outliers at +/-2 standard deviations. Weighted densities were applied to the block model based on the percentage of block volume within each wire-frame. The amount of potential ore material (not rhyolite, voids or overburden) was also assigned to each block for use in mineral resource reporting. Density assignment values are shown in Table 14.8.

Table 14.8: Hangar Flats Density Assignment Values

Rock Model Unit	Bulk Density (g/cm^3)	Bulk Density (lbs/ft^3)
QM/AK/GRAN	2.63	164.2
Meadow Creek Fault	2.60	162.3
11-99 Fault	2.63	164.2
Diabase	2.61	162.9
Gouge	2.55	159.2
Rhyolite	2.54	158.6
Overburden	1.75	109.2

The Hangar Flats mineral resource estimate was completed for gold, antimony and silver using the estimation domains and shells discussed previously. Gold was estimated using ordinary Kriging within the 0.25 g/t Au grade shell in three passes. To mitigate the risk associated with use of historical data, the first pass used only post-1953 drill holes, whereas the second and third passes utilized all data. The influence of pre- vs. post-1953 data was calculated for each block using the same correlogram weighting as used for gold. Pass one for gold estimation was limited to a search based on correlogram ranges at 80% of the sill, the second pass was expanded from the first and the third pass using relaxed sample requirements and ellipse anisotropy. The boundary between Domains 1 and 2 was treated as a soft boundary with no restrictions on composite selection. Composites representing rhyolite, overburden, voids and backfill were not utilized in the estimates. A nearest neighbor estimate was also performed and used for verification purposes. Table 14.9 shows the search and sample selection parameters for the estimates.

Table 14.9: Estimation Parameters for Hangar Flats

Metal	Gold						Silver		Antimony		
Domain/Shell	Domain 1			Domain 2			Domain 1	Domain 2	Inside Sb Shell		Outside Sb Shell
Pass	1	2	3	1	2	3	1	2	1	2	1
Method	OK	OK	OK	OK	OK	OK	OK	OK	OK	IDS	IDS
Principal Axis Azimuth / Plunge(1)	350 / -5	350 / -5	350 / -5	350 / -18	350 / -18	350 / -18	348 / -3	157 / 27	26 / 2	26 / 2	0 / 0
Intermediate Axis Azimuth / Plunge(1)	80 / 6	80 / 6	80 / 6	77 / 9	77 / 9	77 / 9	78 / 3	337 / 63	125 / 77	125 / 77	90 / 0
Minor Axis Azimuth / Plunge(1)	120 / -82	120 / -82	120 / -82	142 / -70	142 / -70	142 / -70	123 / -86	247 / 0	295 / 13	295 / 13	0 / -90
Principal Axis Search Distance (m)	60	100	150	90	130	150	100	150	40	80	50
Major / Intermediate / Minor Axis	1 : 0.20 : 0.40	1 : 0.20 : 0.40	1 : 0.30 : 0.60	1 : 0.55 : 0.45	1 : 0.55 : 0.45	1 : 0.65 : 0.50	1 : 0.35 : 0.65	1 : 0.44 : 0.52	1 : 0.5 : 0.25	1 : 0.5 : 0.25	1 : 1 : 1
Search Type	4-Sector	4-Sector	1-Sector	4-Sector	4-Sector	4-Sector	4-Sector	4-Sector	4-Sector	4-Sector	Spherical, 1-Sector
Composite Restrictions	Domain 1&2, Post-1953	Domain 1&2	Domain 1&2	Domain 1&2, Post-1953	Domain 1&2	Domain 1&2	N/A	N/A	Domain Sb, Post-1953	Domain Sb	Outside
Maximum Composites / Sector	2	2	12	2	2	2	2	3	2	2	8
Minimum Composites	5	3	2	5	3	2	3	4	3	3	3
Minimum No. of Holes	2	2	1	2	2	1	N/A	N/A	2	2	2
Maximum Composites / Hole	4	4	6	4	4	6	N/A	N/A	4	4	4

Note:
(1) Negative plunge is downward.

Antimony was also estimated in three passes. The first pass used ordinary Kriging, and extended to a range of nearly 90% of the sill. The second pass extended the search to twice the first pass range and utilized inverse distance squared weighting for estimation of blocks that were not estimated in the first pass. The third pass estimated blocks outside of the antimony domain by inverse distance squared interpolation with an omni-directional ellipsoid search and isotropic weighting. Pre-1953 data was filtered for use, depending upon pass, in the same manner as the gold estimate.

Silver was estimated using ordinary Kriging within domains 1 and 2, separately. The boundary between the domains was treated as soft for silver estimation. Only blocks that received a gold estimate were estimated for silver.

14.2.9 Block Model Validation

The block model for the Hangar Flats mineral resource was validated by completing a series of graphical inspections, bias checks, sensitivity studies and comparison to prior estimates. Graphically, the model was checked by reviewing the block estimate relative to the geologic model, domain boundaries and grade shell. Block model variables were checked to ensure that they fall within appropriate ranges. Global bias was assessed by comparing the estimated grade to the nearest neighbor estimate. Local bias was assessed on swath plots in the X, Y and Z directions as shown on Figure 14.3, Figure 14.4, and Figure 14.5, respectively.

Change of support for gold was assessed using a Hermitian Correction model (HERCO) and indicates -1% to +3% bias in contained metal for Domains 1 and 2 between the Kriged estimate and the theoretical grade-tonnage distribution at a cutoff grade of 0.75 g/t Au indicating the Kriged estimate yields an appropriate level of smoothing. Relative to the 2012 PEA estimate, the new estimate yields an 18% increase in indicated gold ounces and a 15% increase in antimony pounds, consistent with the addition of new drill holes to the estimation database.

Figure 14.3: North-South Gold Swath Plot for Hangar Flats

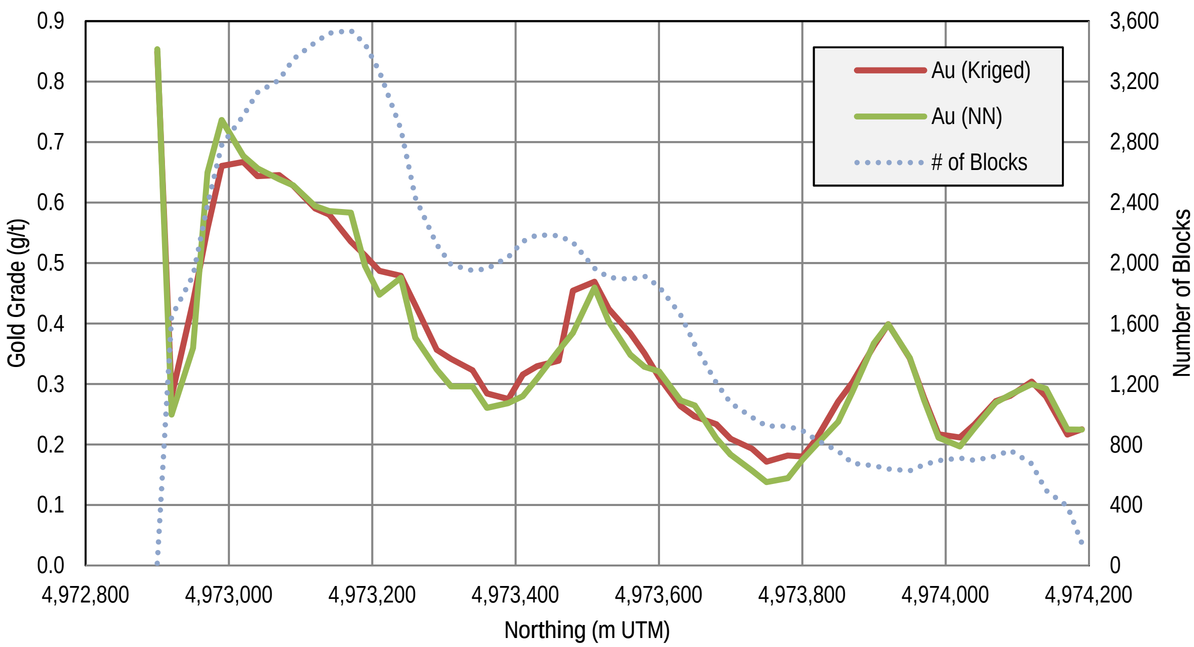


Figure 14.4: East-West Gold Swath Plot for Hangar Flats

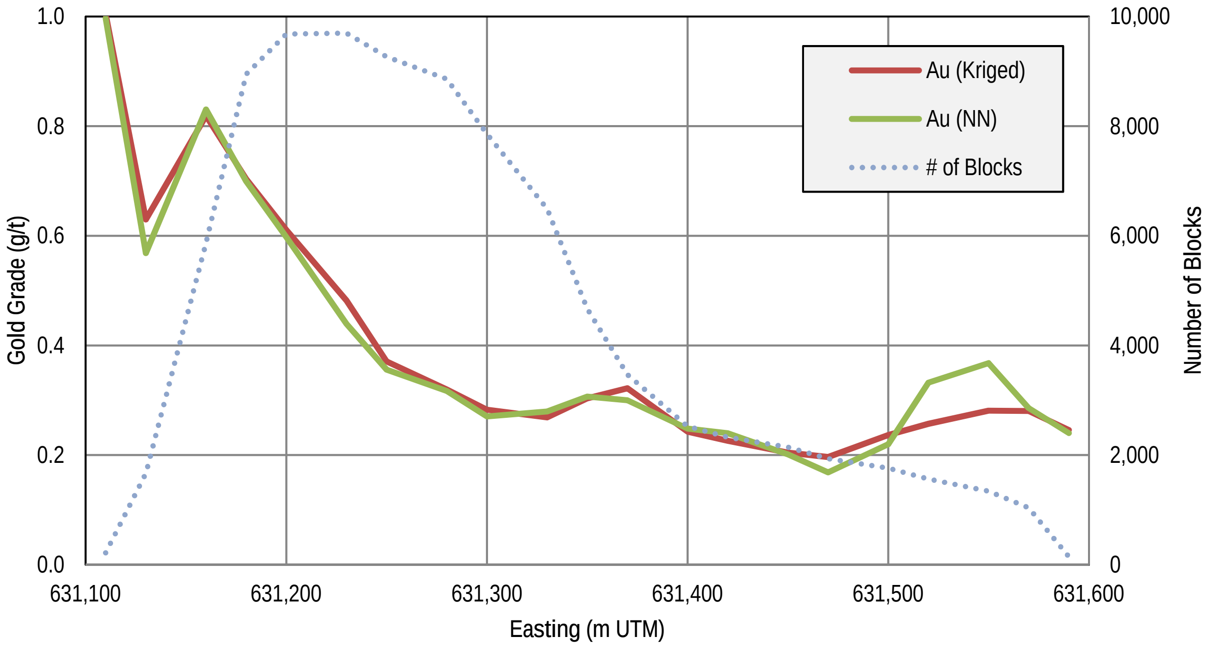
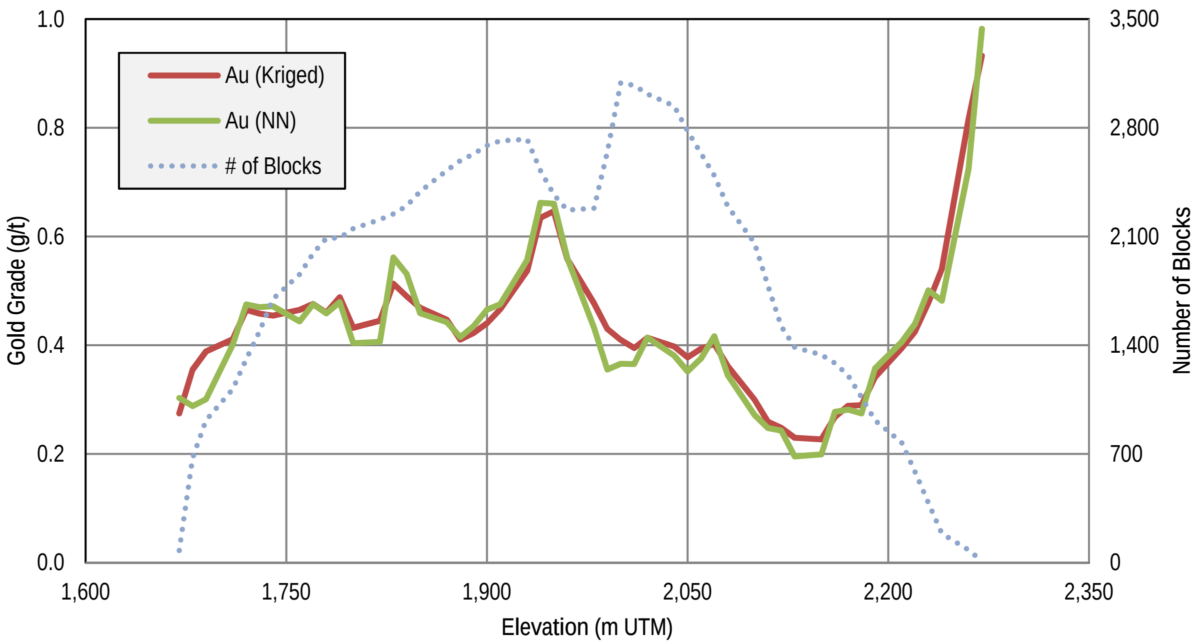


Figure 14.5: Elevation Gold Swath Plot for Hangar Flats



14.2.10 Hangar Flats Mineral Resource Classification

Confidence criteria used to guide mineral resource classification include composite distance, number of drill holes used, number of composites, influence of post-1953 data in the estimate, and single block Kriged results. Blocks eligible for indicated classification for gold are those estimated in the first estimation pass with a minimum anisotropic distance of <50 m, or those estimated in the second pass meeting the above criteria with composites in at least two sectors. Only blocks in Domains 1 or 2 are eligible for indicated classification. Antimony mineral resources require at least three composites from two drill holes with a minimum distance of 40 m to be eligible for indicated classification. Antimony in blocks meeting the gold criteria, but not the antimony criteria, are not included in the antimony mineral resource estimate. Single block Kriged results support the gold classification strategy. Final classification was applied following manual smoothing of the results on 6 m plan sections to produce a model with reasonably contiguous zones of inferred and indicated blocks.

14.3 WEST END

14.3.1 Mineral Resource Estimation Procedures

The West End mineral resource estimation is based on the validated and verified drill hole database, interpreted lithologic units, interpreted fault structures, and LiDAR topographic data. The geologic modeling and estimation of mineral resources was completed using the commercial three-dimensional block modelling and mine planning software packages Geovia GEMS™ 6.6 and Micromine™ Version 14; geostatistics and semi-variogram analyses were completed using Isaaks & Co.'s SAGE2001™ software package.

14.3.2 Drill Hole Database

The drill hole database supplied by Midas Gold for mineral resource modeling included 940 drill holes in Excel format. The database consisted of collar locations in UTM grid coordinates, drill hole orientations with downhole surveys, assay intervals with gold and silver analyses by fire assay and/or cyanide soluble assay, geologic intervals with rock types, core recovery information and specific gravity measurements.

The West End deposit was previously in production as a heap leach operation and many infill drill holes were drilled during the 1980s and 1990s using various methods, as previously described in Section 10. The drill holes were reviewed, and certain drill holes were not considered reliable for use in mineral resource estimation, including rotary and air-track drill holes, and other un-reliable holes flagged by Midas Gold. After removal of selected drill holes and non-bedrock intervals, the final database contained 674 drill holes.

Detection limits for gold are quite variable, depending on the drilling campaign and assay lab used. Detection limits were adjusted to values equal to half the detection limit; levels well below those of economic interest. Approximately 78% of the assay records have gold fire assays (**AuFA**) and 75% have cyanide soluble gold assays (**AuCN**). Some historic operators selectively used fire assays within the sulfide zones where sulfide mineralization was observed, resulting in a dataset that contains some "spot" AuFA records. This results in an apparent high bias because higher-grade intervals were selected for fire assay. To address this, a new variable was created (Au_Final) combining AuFA if available, and AuCN if not, ensuring that an assay is available for every interval in holes containing partial fire assay data. While this treatment is somewhat conservative, it affects a relatively small subset of drill holes in a restricted area of the deposit and as such will not result in over-estimation of *in situ* mineral resources based on selective spot assaying of higher-grade intervals. Au_Final, the variable used for estimation of total gold and discussed in the remainder of this section, is shown by drilling campaign in Table 14.10.

Partial or spot assaying for AuCN is prevalent throughout the deposit, especially within the Superior-era drill holes where available AuCN assays do not adequately define the transition from oxide to sulfide gold. This issue was addressed by removing 70 drill holes with incomplete AuCN assays following section-by-section review for completeness and potential impacts to the mineral resource estimate. The final dataset for estimation of cyanide soluble gold is shown in Table 14.10.

Only Midas Gold, Canadian Superior Mining Ltd. (**Superior**) and Stibnite Mines Inc. (**SMI**) drill holes were assayed for silver, with the latter exclusively assayed for cyanide soluble silver. Similar to the treatment of partial gold assays, a new variable Ag_Final was created combining fire assay and cyanide soluble silver assays for use in silver estimation.

Table 14.10: Drill Hole Information used in the West End Mineral Resource Estimate

Company	Au Fire Assay			Au Cyanide Assay			Silver		
	# Holes	# Samples	Meters	# Holes	# Samples	Meters	# Holes	# Samples	Meters
El Paso	1	18	30	0	0	0	0	0	0
Midas Gold	53	6,020	11,499	52	5,148	9,872	53	6,020	11,499
Pioneer	336	21,313	32,498	336	21,281	32,449	136	6,947	10,586
SMI	118	6,851	10,431	118	6,851	10,431	118	6,851	10,431
Superior	163	6,573	11,626	132	2,850	6,196	71	2,642	5,448
Twin River	3	160	256	0	0	0	0	0	0
All	674	40,935	66,340	638	36,130	58,948	378	22,460	37,964

Note: Drill hole information excludes samples within overburden and includes un-sampled intervals.

Drill holes in the West End deposit form an irregular grid and are primarily vertical or oriented on 120 degree azimuths. Mean drill hole spacing is approximately 40 m above 2,100 m elevation increasing to 70 m near the base of the drill pattern at 1,900 m elevation.

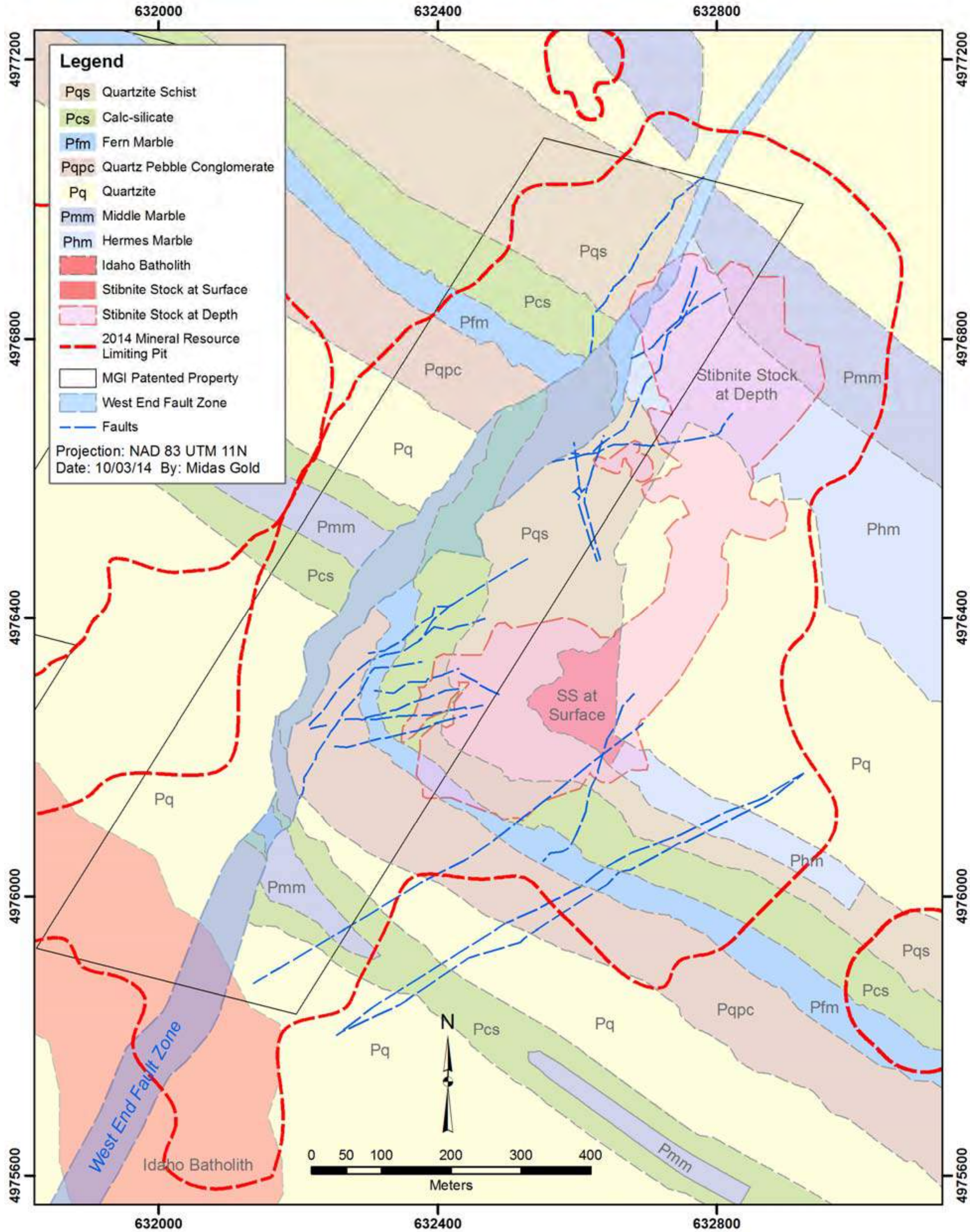
The vast majority of assay lengths are 1.52 m (5 ft) for the historic campaigns and 1.52 m (5 ft) to 2.1 m (7 ft) for the Midas Gold drill holes. The mean sample length is 1.61 m.

14.3.3 Geologic Modeling

The West End deposit occurs in an overturned sequence of steeply dipping Proterozoic to Paleozoic metasediments comprising the Stibnite Roof Pendant. As discussed in Section 7, lithologic units consist of quartzite, quartz-pebble conglomerate, interbedded quartzite and schist, limestones, dolomitic marble, and calc-silicate rocks and range in thickness from 70 – 180 m. The meta-sedimentary rocks are intruded by quartz-monzonite and granitic stocks. Mineralization occurs within and adjacent to fault zones, principally the southeast dipping WEFZ.

The geologic model prepared by Midas Gold consists of eight northeast-dipping lithostratigraphic units which intersect, and are offset across, the WEFZ (Figure 14.6). The WEFZ is modeled as two surfaces representing the hanging wall and footwall of the structural corridor and is up to 80 m wide, dipping 50 to 70 degrees to the southeast with a strike length of over 1.7 km. Additional wireframes representing splay faults and subsidiary structures to the main WEFZ were also provided. The geologic model also includes two intrusive units, the Stibnite Stock of granitic composition which intrudes the metasediments 200 to 300 m east of the WEFZ, and the Idaho Batholith, an intrusion of quartz monzonite composition occurring at the southeast margin of the deposit. Midas Gold provided a topographic surface derived from 1 m gridded LiDAR flown in 2009, a pre-mining topographic surface constructed from historic maps and drill hole collars, and an overburden surface representing the current top of bedrock constructed from drill hole data, and historic pit ‘as-builts’ representing the extent of historic mining.

Figure 14.6: Geologic Model for West End



Gold mineralization in the West End deposit occurs within all lithostratigraphic units with higher-grade mineralization preferentially occurring in the schist and calc-silicate lithologies. Gold mineralization is associated with silica alteration occurring as quartz-veinlets, stockworks and zones of silica flooding and replacement. Gold also occurs along oxidized fractures and broadly disseminated within fracture zones and within intrusive units where gold is associated with sulfide-sericite alteration. Gold is concentrated along and adjacent to the WEFZ and its subsidiary structures; with mineralized drill holes observed crossing the modeled hanging wall and footwall with no apparent disruptions in gold grade. Silver mineralization within the deposit is generally low-grade and erratic. Silver mineralization is locally elevated within the WEFZ.

The oxidation level in the deposit is of moderate and variable depth, with elevated AuCN values occurring at shallow levels, preferentially within certain lithologic units, and locally at deeper elevations between strands of the WEFZ and along splay structures. AuCN mineralization is only sparsely tested below the 1,900 m elevation in widely spaced Midas Gold drill holes.

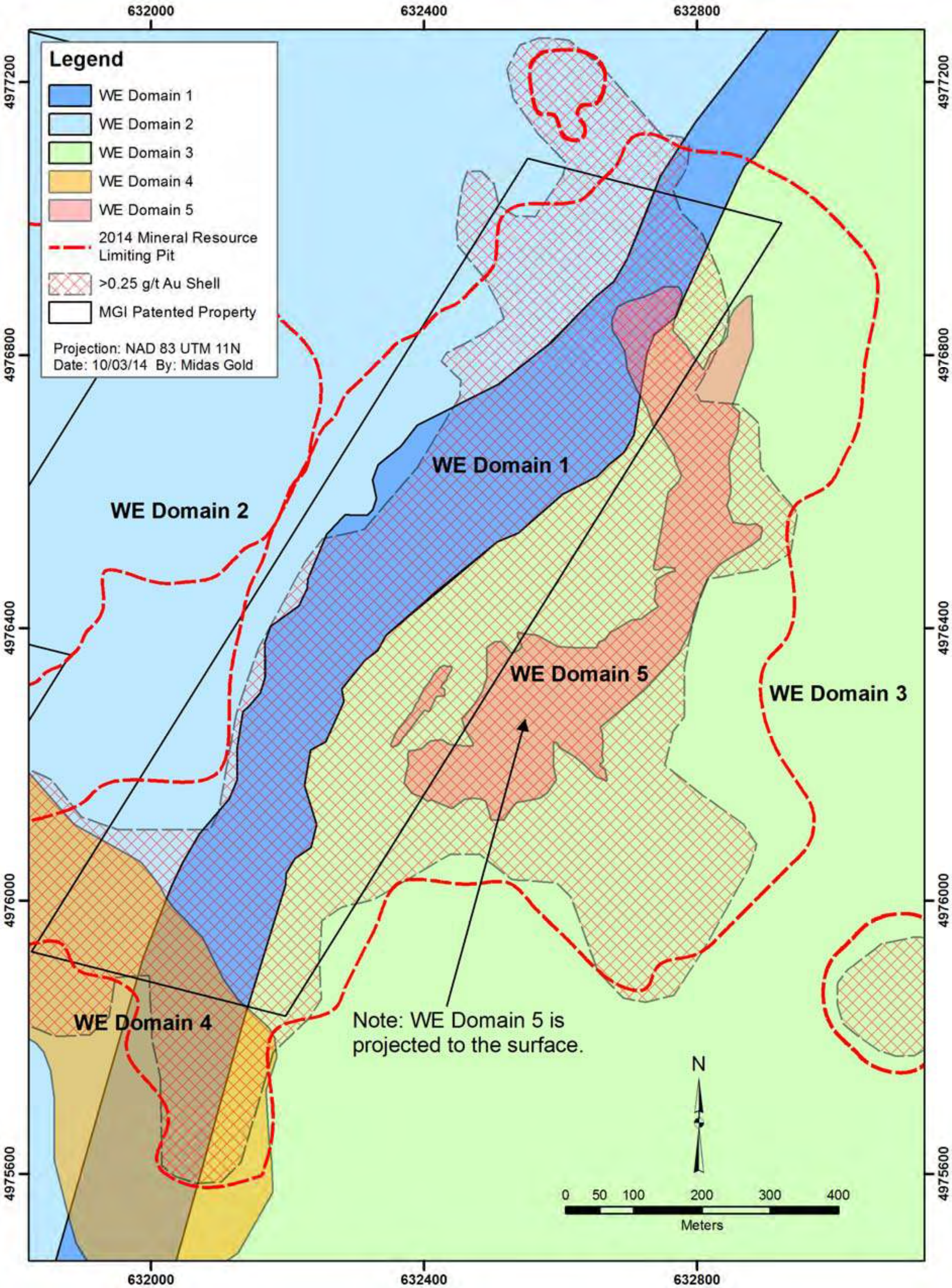
14.3.4 Estimation Domain Modeling

The West End gold estimates utilized the geologic model, a 0.25 g/t Au grade shell and five estimation domains to characterize gold deposition in relation to structural and stratigraphic controls. Boundary treatment during estimation is based on analysis of grade variability across geologic and estimation domain contacts.

A grade shell was constructed based on an indicator estimate using a cutoff grade of 0.25 g/t Au_Final. The shell demonstrates reasonable continuity along strike and vertically. For AuCN, the shell was modified slightly and restricted to elevations above 1,911 m. A grade shell was not developed for silver because mineralization is erratic and generally low grade throughout the deposit.

Estimation domains for Au_Final and AuCN were developed from the geologic model, perceived structural controls on mineralization, grade contouring, graphical plots of assay data and the 0.25 g/t Au grade shells. The WEFZ exerts a strong structural control on gold mineralization but fault contacts are not distinct boundaries; rather gold mineralization extends into favorable stratigraphic units adjacent to the structure. Estimation Domain 1 is the WEFZ expanded 50 m to the east and west, so as to encompass mineralization within and adjacent to the fault zone. Domains 2 and 3 include mostly low-grade material to the west and east of the WEFZ respectively, exclusive of Domains 4 and 5. Domain 4 encompasses generally low-grade gold mineralization hosted primarily within quartz monzonite of the Idaho batholith in the southwest region of the deposit. Domain 5 is entirely within Domain 3 and captures gold mineralization associated with east-northeast striking splay faults within the Stibnite Stock in the eastern region of the deposit. The West End estimation domains are shown on Figure 14.7.

Figure 14.7: Estimation Domains for West End



Stratigraphic controls exert a strong influence on gold mineralization and were investigated within estimation domains through the use of descriptive statistics. In the simplified geologic model, certain lithostratigraphic units encompass multiple rock types (i.e. the quartzite-schist formation) and statistics were prepared for both logged lithology groupings and for samples occurring within modeled geologic units. In general, the distribution of gold within geologic domains is similar to that observed in logged lithology types, with the highest mean gold grades observed within the quartzite-schist formation (quartzite, psammite and schist), the calc-silicate formations (marble, metapelite, calcareous schists) and fault related rocks (breccia and gouge) (Table 14.11). The carbonate units (marble, limestone and dolomite) and clastics (conglomerate and quartzite) are generally lower grade.

Table 14.11: West End Descriptive Statistics by Rock Solid

Statistic	Rock Solid (g/t Au)											
	QZ	Hermes Marble	Upper Quartzite	Quartzite Schist	LCS	FM	QPC	Lower Quartzite	MM	UCS	ID Batholith	Stibnite Stock
Mean	0.518	0.290	0.585	1.114	1.044	0.619	0.645	0.358	0.970	0.439	0.452	0.605
Standard Error	0.031	0.032	0.044	0.019	0.024	0.027	0.017	0.015	0.098	0.039	0.072	0.016
Median	0.170	0.137	0.274	0.4625	0.274	0.137	0.240	0.156	0.309	0.081	0.171	0.343
Standard Deviation	0.909	0.766	0.799	1.754	2.064	1.441	1.158	0.656	1.933	1.031	1.348	0.914
Minimum	0.0025	0.0025	0.0025	0.0025	0.0025	0.0025	0.0025	0.0025	0.0100	0.0025	0.0050	0.0025
Maximum	8.503	11.52	6.000	22.63	28.22	17.62	16.11	9.394	13.61	12.55	20.91	16.63
Count	877	567	327	8,536	7,587	2,894	4,437	1,832	392	686	347	3,251
CV	1.75	2.64	1.36	1.57	1.97	2.32	1.79	1.83	1.99	2.35	2.98	1.50

Variations in mean grade of gold assays across geologic boundaries were examined using contact plots to determine sub-domains requiring hard-boundary treatment during estimation. In Domain 1, abrupt grade changes were noted between the Hermes Marble and Quartzite-Schist formations and between the Lower Calc-silicate and Lower-quartzite formations. Within Domain 1, the resulting estimation sub-domains separate the Hermes Marble and the Lower Quartzite from other units. AuCN is generally comparable to gold within the Au_Final domains, with the exception of Domain 3 where the Lower Calc-silicate is a sub-domain with 10 m soft boundaries. Boundary conditions of the estimation domains for Au_Final were examined using contact plots and analysis of composited data (discussed below) and indicate both hard and soft boundaries between various estimation domains and lithologic sub-domains which were applied where warranted. For silver, marble units in Domain 1 form a distinctly lower-grade population and were estimated separately from other lithologies. Otherwise, the silver estimation domains are the same as those for gold but do not segregate geologic solids into sub-domains.

14.3.5 Compositing

Gold, AuCN and silver assays were composited downhole on 3 m intervals across geologic and estimation domain boundaries and excluding un-assayed or missing intervals. Composites <1 m were removed from the final data set. The 3 m composite length is an even multiple of the majority of raw assay lengths and represents 50% of the proposed mining bench height and estimation block height.

Table 14.12: West End Raw Composite Statistics by Estimation Domain

Statistic	Domain 1			Domain 2		Domain 3		Domain 4	Domain 5
	HM	LQ	Other	LCS/QZ_SCH	Other	LCS	Other	ID Batholith	Stibnite Stock
Fire Assay Statistics (g/t Au)									
Mean	0.335	0.491	1.179	0.751	0.414	1.046	0.508	0.445	0.493
Standard Error	0.062	0.023	0.017	0.049	0.032	0.041	0.017	0.039	0.013
Median	0.183	0.313	0.644	0.388	0.250	0.337	0.224	0.273	0.293
Standard Deviation	0.847	0.617	1.483	1.007	0.494	2.025	0.898	0.605	0.651
Minimum	0.004	0.003	0.003	0.003	0.014	0.003	0.003	0.017	0.017
Maximum	10.29	5.504	18.39	8.503	3.974	22.03	14.36	5.113	10.16
Count	184	702	7,325	426	235	2,445	2,938	239	2,459
Coefficient of Variation	2.53	1.26	1.26	1.34	1.19	1.93	1.77	1.36	1.32
Statistic	Domain 1			Domain 2		Domain 3		Domain 4	Domain 5
	HM	LQ	Other	All		LCS	Other	ID Batholith	Stibnite Stock
Cyanide Assay Statistics (g/t Au)									
Mean	0.129	0.364	0.661	0.309		0.608	0.344	0.377	0.354
Standard Error	0.014	0.022	0.013	0.020		0.026	0.012	0.040	0.010
Median	0.079	0.238	0.284	0.145		0.179	0.172	0.206	0.209
Standard Deviation	0.176	0.506	1.016	0.493		1.294	0.604	0.533	0.494
Minimum	0.015	0.015	0.015	0.015		0.015	0.015	0.017	0.015
Maximum	1.577	5.143	12.27	4.983		19.46	12.93	4.860	7.948
Count	161	520	5,721	629		2,388	2,730	178	2,410
Coefficient of Variation	1.37	1.39	1.54	1.60		2.13	1.76	1.41	1.39

14.3.6 Evaluation of Outliers

To mitigate estimation risk associated with use of high-grade statistical outliers, capping grades were selected for each estimation domain after declustering and weighting raw 3 m composite data. Capping grade was evaluated using log probability plots and through analysis of contained metal within deciles and centiles, following the Parrish Method (Parrish, 1997). Both methods yielded similar results, except for Domain 1, where the more conservative composite capping grade of 10 g/t Au was selected. AuCN was capped in a manner similar to gold. Silver assays were capped at 10 g/t Ag prior to compositing on 3 m intervals. Table 14.13 shows final capping grades for West End. Low capping grades in domains 2, 4 and 5 are warranted due to low average grade and small relative standard deviation.

Table 14.13: Capping Grades for West End

Statistic	Domain 1	Domain 2	Domain 3	Domain 4	Domain 5
Fire Assay Statistics within Gold Shell (g/t Au)					
Number	8,211	661	5,383	239	2,459
Maximum Value	18.39	8.503	22.03	5.113	10.16
Cap Value	10	3	10	2.3	3
Number Capped	14	24	33	3	27
Mean Uncapped	1.101	0.631	0.753	0.445	0.493
Mean Capped	1.096	0.593	0.731	0.420	0.477
Metal removed (%)	0.46	6.41	2.92	6.05	3.43

Statistic	Domain 1	Domain 2	Domain 3	Domain 4	Domain 5
Cyanide Assay Statistics within Gold Shell (g/t Au)					
Number	6,402	629	5,118	178	2,410
Maximum Value	12.27	4.983	19.46	4.860	7.948
Cap Value	6	4	9	4	3
Number Capped	30	2	12	1	12
Mean Uncapped	0.624	0.309	0.467	0.377	0.354
Mean Capped	0.616	0.307	0.460	0.372	0.345
Metal removed (%)	1.20	0.80	1.50	1.30	2.57

Table 14.14: West End Descriptive Statistics for Capped Composites

Statistic	Domain 1			Domain 2		Domain 3		Domain 4	Domain 5
	HM	LQ	Other	LCS/QZ_SCH	Other	LCS	Other	ID Batholith	Stibnite Stock
Fire Assay Statistics within Gold Shell (g/t Au)									
Mean	0.333	0.491	1.173	0.695	0.410	1.003	0.505	0.420	0.477
Standard Error	0.061	0.023	0.017	0.038	0.030	0.035	0.016	0.029	0.011
Median	0.183	0.313	0.644	0.388	0.250	0.337	0.224	0.273	0.293
Standard Deviation	0.828	0.617	1.440	0.777	0.467	1.733	0.863	0.456	0.528
Minimum	0.004	0.003	0.003	0.003	0.014	0.003	0.003	0.017	0.017
Maximum	10.00	5.504	10.00	3.000	3.000	10.00	10.00	2.300	3.000
Count	184	702	7,325	426	235	2,445	2,938	239	2,459
Coefficient of Variation	2.49	1.26	1.23	1.12	1.14	1.73	1.71	1.09	1.11
Statistic	Domain 1			Domain 2		Domain 3		Domain 4	Domain 5
	HM	LQ	Other	All		LCS	Other	ID Batholith	Stibnite Stock
Cyanide Assay Statistics within Gold Shell (g/t Au)									
Mean	0.129	0.364	0.653	0.307		0.595	0.342	0.372	0.345
Standard Error	0.014	0.022	0.013	0.019		0.024	0.011	0.037	0.008
Median	0.079	0.238	0.284	0.145		0.179	0.172	0.206	0.209
Standard Deviation	0.176	0.506	0.958	0.472		1.173	0.578	0.495	0.411
Minimum	0.015	0.015	0.015	0.015		0.015	0.015	0.017	0.015
Maximum	1.577	5.143	6.000	4.000		9.000	9.000	4.000	3.000
Count	161	520	5721	629		2388	2730	178	2410
Coefficient of Variation	1.37	1.39	1.47	1.54		1.97	1.69	1.33	1.19

14.3.7 Statistical Analysis and Spatial Correlation

Correlogram models were developed using the SAGE2001™ software package to guide the search ellipses and establish spatial correlation and sample weighting for the estimates. The nugget effect was derived using down-hole correlograms. Correlograms for capped 3 m composites within the grade shell were developed for Au_Final, AuCN and Ag_Final within estimation Domains 1, 2 and 3. The intrusive rocks in Domains 4 and 5 did not yield reliable correlograms and were estimated using inverse distance weighting. Gold and cyanide soluble gold demonstrate spatial continuity along dominant structural trends and within stratigraphic units. Cyanide soluble gold demonstrates greater vertical continuity than Au_Final. The correlogram parameters are summarized in Table 14.15.

Table 14.15: Correlogram Models for West End

Element	Estimation Domain	Ellipse Axes Azimuth/Plunge 1st	2nd	3rd	Nugget C0	Sill C1/C2	Modeled Ranges Model 1 / Model 2 1st	2nd	3rd	Type
Au_Final	1 & 2	254/-47	4/-17	108/-38	0.067	0.601/0.332	16/275	16/82	20/43	Exp
	3	95/-33	45/44	166/27	0.186	0.814	62	33	14	Exp
	4 & 5	Inverse Distance Weighting								
Au_CN	1 & 2	49/-20	142/-6	67/69	0.161	0.839	101	50	35	Exp
	3 - LCS	115/-53	44/14	144/33	0.099	0.901	47	27	14	Exp
	3 - Other	63/-6	175/-75	151/14	0.15	0.85	41	21	10	Exp
	4 & 5	Inverse Distance Weighting								
Ag	1	86/-22	353/-6	69/67	0.234	0.766	86	64	52	Exp
	2 & 3	37/-38	77/44	325/21	0.039	0.961	164	37	14	Exp
	4 & 5	Inverse Distance Weighting								

14.3.8 Block Model Parameters and Grade Estimation

The West End block model comprises 15.24 x 15.24 x 6.096 m blocks (50 x 50 x 20 ft) with coordinates defined in Table 14.16. Blocks were discretized into a 5 x 5 x 2 array of points during estimation.

Table 14.16: Block Model Definition for West End

Deposit	Dimension (m) X	Y	Z	Origin (m) X	Y	Z	Number of Blocks X	Y	Z	Rotation
West End	15.24	15.24	6.096	631,727.6	4,975,408.0	1,704.0	97	127	114	0
Note: Block centroid, NAD83 Zone 11N Datum										

The drill hole database contains 166 density measurements from the primary lithologic units, the majority of which were determined onsite using the water immersion method, with a number of independent third party measurements completed offsite using the same methodology. Because of the relatively small number of density measurements, density values were averaged for each lithologic unit and assigned to the geologic model after removal of outliers, as summarized in Table 14.17.

Table 14.17: West End Density Assignment Values

Rock Model Unit	Bulk Density (g/cm^3)	Bulk Density (lbs/ft^3)
Quartzite & Background	2.61	162.9
Quartzite-Schist	2.70	168.6
Lower Calc-Silicate	2.74	171.1
Fern Marble	2.78	173.5
Qtz Pebble Conglomerate	2.63	164.2
Lower Quartzite	2.65	165.4
Middle Marble	2.80	174.8
Upper Calc-Silicate	2.76	172.3
ID Batholith	2.54	158.6
Stibnite Stock	2.61	162.9
Overburden	1.75	109.2

The West End mineral resource estimate was completed for gold, cyanide-soluble gold and silver using the estimation domains and sub-domains discussed previously. The estimate is limited to blocks occurring within the

0.25 g/t Au_Final grade shell below the current LiDAR topographic surface. Estimates for Au_Final, AuCN and Ag_Final in Domains 1, 2 and 3 are derived by ordinary Kriging using the 3 m composite file and correlogram weighting models discussed above. Estimates for Domains 4 and 5 use inverse-distance weighting to various powers. Estimation is performed in two passes; the first pass is limited to a search based on the correlogram ranges at approximately 90% of the sill. The second pass search is expanded to a multiple of the first pass with greater isotropy. Composites occurring above the current bedrock surface from pre-historic mining activity are utilized in the estimate, but material in this region is not included in the mineral resource. Table 14.18 shows estimation criteria for gold and cyanide soluble gold by domain. Silver was estimated in a manner similar to gold but is not detailed in this Report because it is of minor economic significance.

Table 14.18: Summary of Estimation Parameters for West End

Parameter	1			2		3		4	5
	HM	LQ	Other	LCS/ QZ_SCH	Other	LCS	Other	ID Batholith	Stibnite Stock
Pass 1 – Fire Assay Au									
Pass 1 Method	OK	OK	OK	OK	OK	OK	OK	ID3	ID2
Princ. Axis Az/Plunge(1)	280/-35	280/-35	280/-35	280/-35	280/-35	45/-48	45/-48	0/0	0/0
Int. Axis Az/Plunge	346/30	346/30	346/30	346/30	346/30	93/31	93/31	0/-90	0/-90
Minor Axis Az/Plunge	226/40	226/40	226/40	226/40	226/40	347/-25	347/-25	90/0	90/0
Princ. Axis Srch Dist. (m)	50	50	50	40	40	50	50	40	50
Maj/Int/Minor Axis	1:.67:.55	1:.67:.55	1:.67:.55	1:.67:.55	1:.67:.55	1:.65:.35	1:.65:.35	01:01:01	01:01:01
Search Type	Sector	Sector	Sector	Open	Open	Sector	Sector	Sector	Sector
Comp Restrictions	Hard	Hard	Hard	Hard	Hard	Soft (10m)	Soft (10m)	Hard	Hard
Max Comps/Sector	3	3	3	12	12	3	4	4	3
Min Comps	4	4	4	3	3	4	5	5	4
Min # of Holes	2	2	2	2	2	2	2	2	2
Max Comps/Hole	5	5	5	4	5	5	4	4	4
Pass 2 – Fire Assay Au									
Pass 2 Method	OK	OK	OK	OK	OK	OK	OK	ID3	ID2
Princ. Axis Az/Plunge	280/-35	280/-35	280/-35	280/-35	280/-35	356/2	356/2	0/0	0/0
Int. Axis Az/Plunge	346/30	346/30	346/30	346/30	346/30	79/-75	79/-75	0/-90	0/-90
Minor Axis Az/Plunge	226/40	226/40	226/40	226/40	226/40	267/-15	267/-15	90/0	90/0
Princ. Axis Srch Dist. (m)	150	150	150	150	150	150	150	75	150
Maj/Int/Minor Axis	1:.67:.55	1:.67:.55	1:.67:.55	1:.75:.75	1:.67:.55	1:.7:.4	1:.7:.4	01:01:01	01:01:01
Search Type	Open	Open	Open	Open	Open	Sector	Sector	Sector	Sector
Comp Restrictions	Hard	Hard	Hard	Hard	Hard	Soft (10m)	Soft (10m)	Hard	Hard
Max Comps/Sector	12	12	12	12	12	3	3	4	2
Min Comps	2	2	2	2	2	2	2	2	3
Min # of Holes	1	1	1	1	1	1	1	1	1
Max Comps/Hole	5	5	3	4	4	4	4	4	4

Parameter	1			2		3		4	5
	HM	LQ	Other	LCS/ QZ_SCH	Other	LCS	Other	ID Batholith	Stibnite Stock
Pass 1 – Cyanide Assay Au									
Pass 1 Method	OK	OK	OK	OK		OK	OK	IDS	IDS
Princ. Axis Az/Plunge(1)	49/20	49/20	49/20	49/20		115/53	63/6	0/0	57/-1
Int. Axis Az/Plunge(1)	141/6	141/6	141/6	141/6		145/-33	176/75	90/0	144/73
Minor Axis Az/Plunge(1)	247/69	247/69	247/69	247/69		226/15	332/14	90/90	327/17
Princ. Axis Srch Dist. (m)	65	65	90	65		50	50	65	150
Maj/Int/Minor Axis	1:.5:.35	1:.5:.35	1:.5:.35	1:.5:.35		1:.30:.57	1:.51:.24	01:01:01	1:.64:.36
Sectors	4	4	4	4		4	4	1	4
Domains Permitted	1,3	1,2,3	1,2,3	1,2		1,3	1,3	4	5
Max Comps/Sector	2	2	2	2		3	2	8	2
Max Comps/Sector	2	2	2	2		3	2	8	2
Min Comps	3	3	3	3		2	3	2	3
Min # of Holes	1	1	1	1		1	1	1	1
Max Comps/Hole	4	4	4	4		4	4	4	4

Notes:
(1) Negative plunge is downward.

14.3.9 Block Model Validation

The block model for West End was validated by completing a series of graphical inspections, bias checks, sensitivity studies and comparison to prior estimates. Graphically, the model was checked by reviewing the block estimate relative to the geologic model, domain boundaries and grade shell. Block model variables were check to ensure that they fall within appropriate ranges. Global bias was assessed by comparing the estimated grade to the nearest neighbor estimate within each estimation domain. Local bias was assessed on swath plots in the X, Y and Z directions on Figure 14.8, Figure 14.9, and Figure 14.10, respectively. Change of support was assessed using a HERCO. The HERCO validation suggests that the Kriged models are under-smoothed with respect to the theoretical grade-tonnage distribution at a cutoff grade of 0.75 g/t Au and classification is restricted to the indicated and inferred classes accordingly. A sensitivity model run to assess the results of capping indicates that capping results in an approximate 2% decrease in grade and 2% decrease in tonnage resulting in a 4% decrease in contained gold. Relative to the 2012 PEA mineral resource estimate, the new model indicates a small increase in indicated mineral resources and a 44% decrease in inferred gold ounces. The overall decrease in inferred gold ounces resulted from conversion of inferred mineral resources to the indicated category. Although new drilling successfully expanded indicated mineral resources through conversion of inferred mineral resources to indicated in some areas, this increase was largely offset by the loss of previously indicated material in other areas due to different treatment of the high-biased selective gold fire assays, where un-assayed total gold intervals were not populated with background CN assay data in 2012 but were in 2014.

Figure 14.8: East-West Gold Swath Plot for West End

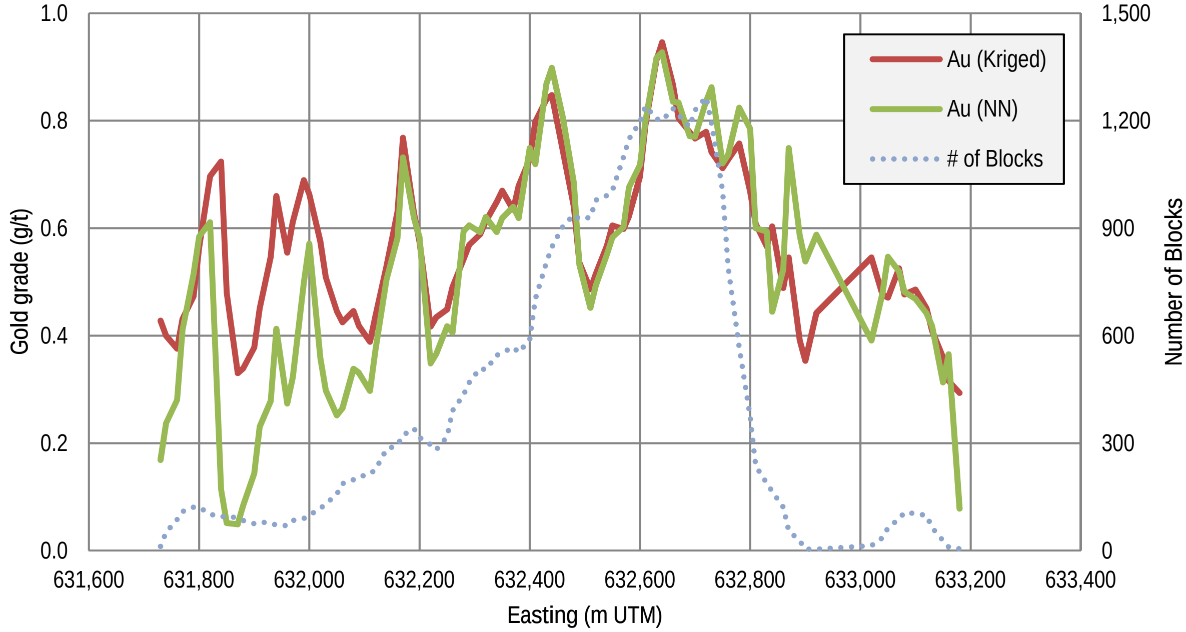


Figure 14.9: North-South Gold Swath Plot for West End

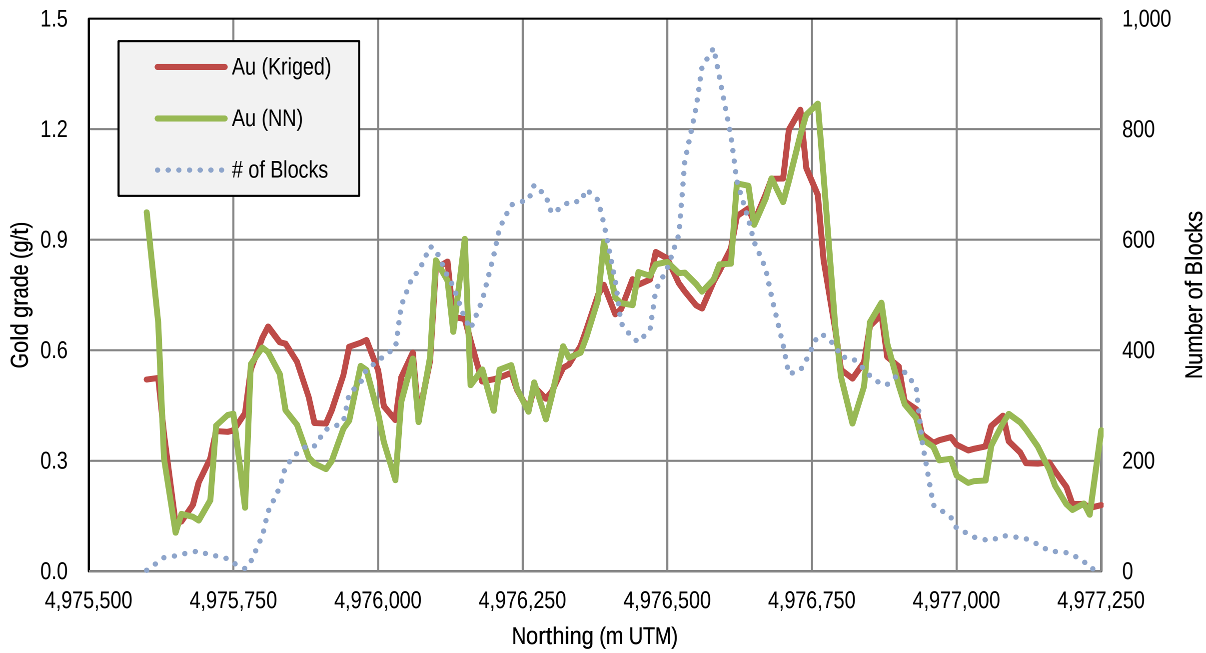
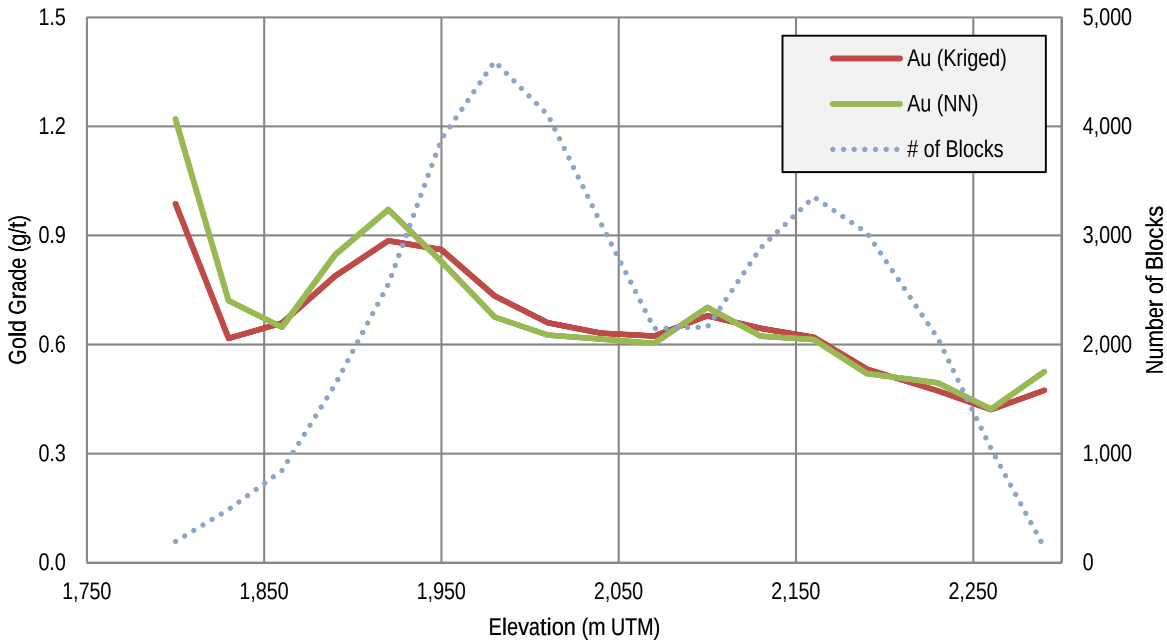


Figure 14.10: Elevation Validation Gold Swath Plot for West End



14.3.10 West End Mineral Resource Classification

Confidence criteria used to guide mineral resource classification include search and composite selection, estimation variance and single-block Kriged results. Blocks eligible for indicated classification are those estimated in the first estimation pass, or those estimated in the second pass by more than 5 composites from two or more drill holes with a Kriging variance <0.45. Distance is not a criteria, but indicated blocks estimated in the second pass generally have an average distance to samples of <50 m and at least one composite within 50 m anisotropic distance. Blocks in Domain 4 are not eligible for indicated classification. Single block Kriged results indicate that the 25 to 70 m drill spacing is sufficient for estimation of quarterly production grade with 85% confidence, suitable for classification of indicated mineral resources. Final classification was applied following manual smoothing of the results on 6 m plan sections to produce a model with reasonably contiguous zones of inferred and indicated blocks.

14.4 YELLOW PINE

14.4.1 Mineral Resource Estimation Procedures

The Yellow Pine mineral resource estimate is based on the validated and verified drill hole database, digitized as-built data of historic workings, interpreted fault structures, sulfide mineralization, and LiDAR topographic data. The geologic modeling and estimation of mineral resources was completed using the commercial three-dimensional block modelling and mine planning software Geovia GEMS™ Version 6.6; geostatistics and semi-variogram analyses were completed using Snowden Supervisor™ Version 8.2 software.

14.4.2 Drill Hole Database

The drill hole database, supplied by Midas Gold in Excel format, contained collar locations surveyed in UTM grid coordinates, drill hole orientations with downhole surveys, assay intervals with gold, antimony, and silver analyses by fire assay and/or cyanide soluble assay, geologic intervals with rock types and specific gravity measurements.

The Yellow Pine deposit was previously in production in the 1930s - 1950s from the so-called glory hole area, while the Homestake area was in production in the 1980s; the entire Yellow Pine area was explored for gold and antimony by numerous operators, up to and including Midas Gold in 2013. The drill hole database contains data for 1,004 separate drill holes representing a mixture of pre-1953 and modern drilling programs. Historical data (i.e. pre-Midas Gold) accounts for approximately 49% of the drill hole database by meterage, as previously described (Section 10). Multiple statistical validations were completed to assess the quality of the historical drill hole data, as discussed in Section 12. A significant number of historic holes were removed from the dataset used for estimation including holes missing critical supporting information, holes with long downhole composited assays, air-track drill holes and all historic pre-1953 drill holes in the northeast portion of the deposit (which is often referred to as the Homestake domain).

For the Yellow Pine deposit, antimony and silver mineral resources were calculated in addition to gold. Table 14.19 shows the number of drill holes and assay intervals utilized in the estimate, which illustrates that the metal values for gold, antimony, and silver were not consistently analyzed for all sample intervals throughout the various historic drilling campaigns. While there are some areas of oxidation within the Yellow Pine deposit, all mineralization is treated as sulfide mineralization with respect to the estimation.

Table 14.19: Drill Hole Information used in the Yellow Pine Mineral Resource Estimate

Company	Gold			Silver			Antimony		
	# Holes	# Samples	Meters	# Holes	# Samples	Meters	# Holes	# Samples	Meters
Barrick	17	2,528	3,909	17	2,528	3,909	17	2,528	3,909
Bradley	109	4,256	6,796	109	4,256	6,796	0	0	0
El Paso	1	60	122	1	60	122	1	60	122
Hecla	67	2,348	3,723	67	2,348	3,723	0	0	0
Midas Gold	226	23,271	43,190	226	23,271	43,190	226	23,271	43,190
Pioneer	1	76	116	1	76	116	1	76	116
Ranchers	145	4,713	7,542	145	4,713	7,542	145	4,713	7,542
Superior	16	393	595	16	393	595	16	393	595
USBM	51	2,828	4,421	51	2,828	4,421	51	2,828	4,421
All	633	40,473	70,414	633	40,473	70,414	457	33,869	59,895
Note: Drill hole information includes un-assayed intervals, and excludes samples in overburden.									

The most common assay lengths are approximately 1.5 m long with the majority of assays between 0.8 m and 2.5 m in length. The drill hole database contains 1,762 specific gravity measurements, collected on core samples using a water immersion method and verified with independent, third party laboratory measurements.

14.4.3 Geologic Modeling

The Yellow Pine mineral resource estimate is based on a generalized geologic model consisting of major rock types, major structures, surfaces, historic underground workings and grade shells for gold, antimony and silver (as shown on Figure 14.11). Intrusive rocks types in the geologic model include the primary host rocks, i.e. quartz-monzonite and granite, which are cut by late-stage diabase and latite dikes. As discussed in Section 7, mineralization in the Yellow Pine deposit is structurally controlled and localized by the MCFZ, a generally north to northeast striking, steeply west-northwest dipping fault zone, and north striking gently west dipping conjugate splay or cross structures associated with the MCFZ. The majority of mineralization in the deposit occurs west of the MCFZ and east of the Hidden Fault Zone (HFZ), a wide, moderately northwest dipping fault and fracture zone. To the south, gold mineralization occurs within and adjacent to the MCFZ, and east of the Hanging Wall Fault (HWF). In the geologic model, the MCFZ and HFZ are modeled as structural corridors containing a variety of fault related rock types including breccia, gouge, cataclasite and rubble zones. To the east of the MCFZ are metasediments of the Stibnite roof pendant, which are not sub-divided in the geologic model. The geologic model also includes solids representing minor late-stage dikes, numerous adits, drifts and underground development workings and surfaces representing current and pre-mining topography and the current top-of bedrock surface. The surface representing the top of bedrock was digitized from drill hole data and from 1950s and 1990s engineering drawings depicting the historical Yellow Pine and Homestake pit bottoms, prior to backfilling. Drill data used to construct the top of bedrock surface includes holes drilled from barges through the pit lake by the Rancher's Exploration Company (Ranchers).

14.4.4 Estimation Domain Modeling

The mineral resource estimate for the Yellow Pine deposit utilized grade shells for gold, antimony and silver in five estimation domains. The gold grade shell was constructed manually using a 0.25 g/t grade threshold. Contouring was controlled by grade values and the geologic and structural trends, particularly the MCFZ, HFZ and HWF. Grade shell construction was limited to no more than 60 m beyond any mineralized drill hole intercept. During interpolation the gold shell served as a hard boundary. The grade shells for antimony and silver were constructed using a similar procedure to the gold grade shell using thresholds of 0.1% Sb and 10 g/t Ag. Both antimony and silver shells are located entirely within the gold grade shell and served as hard boundaries for estimation.

The deposit was divided into five estimation domains to segregate regions with different structural controls on gold mineralization, as indicated by oriented core structural measurements, Midas Gold grade contouring, the geologic model, historical underground mapping and historical reports. Estimation domains are shown on Figure 14.12 and subdivide the deposit into the southern, central and northern regions which show progressively more shallow dipping controls on mineralization, and segregate the MCFZ, HFZ and meta-sedimentary units. The boundaries between the estimation domains were treated as soft boundaries during estimation. Descriptive statistics for raw assays within the gold grade shell are shown in Table 14.20.

Figure 14.11: Geologic Model for Yellow Pine

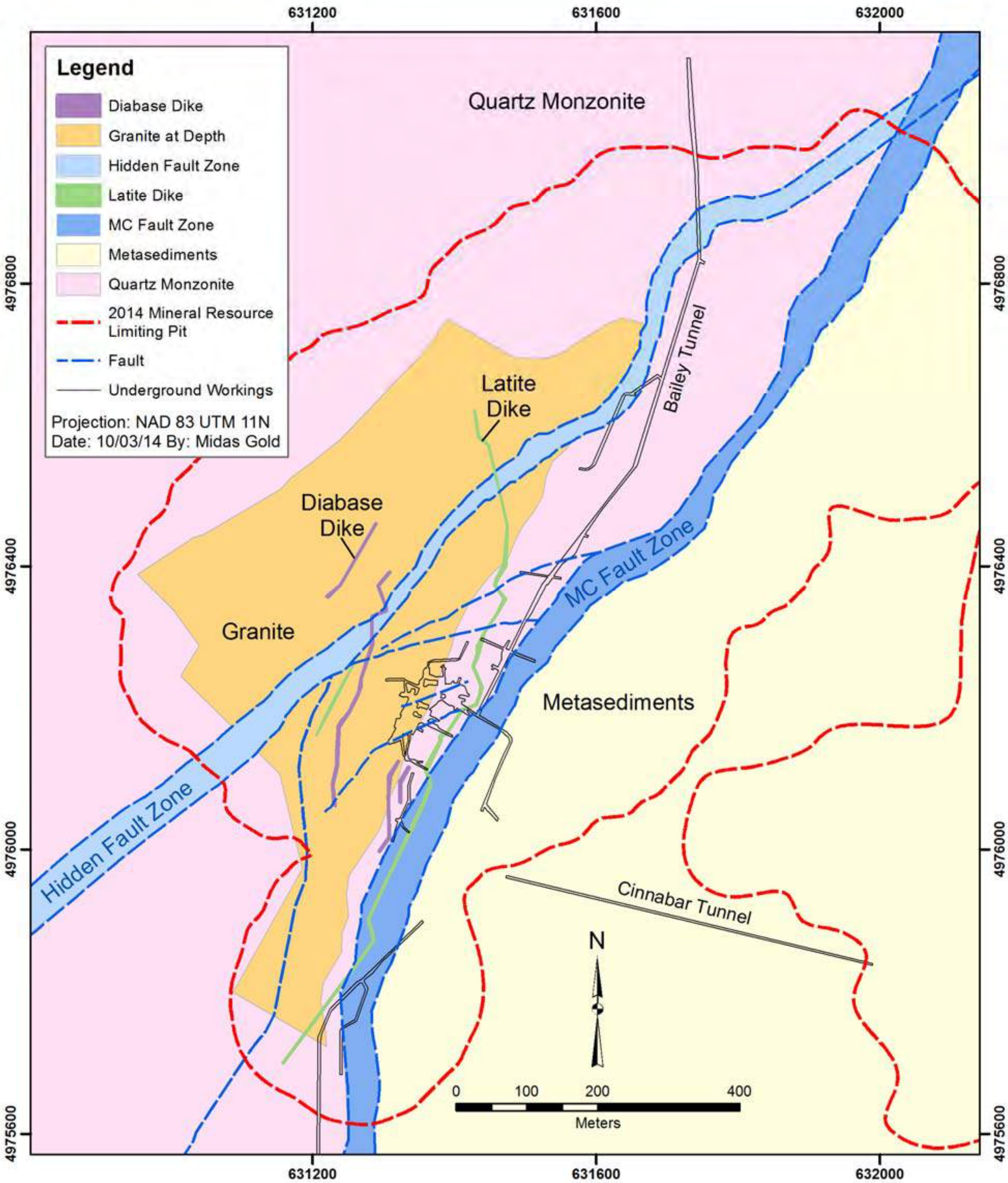


Figure 14.12: Estimation Domains and Grade Shells for Yellow Pine

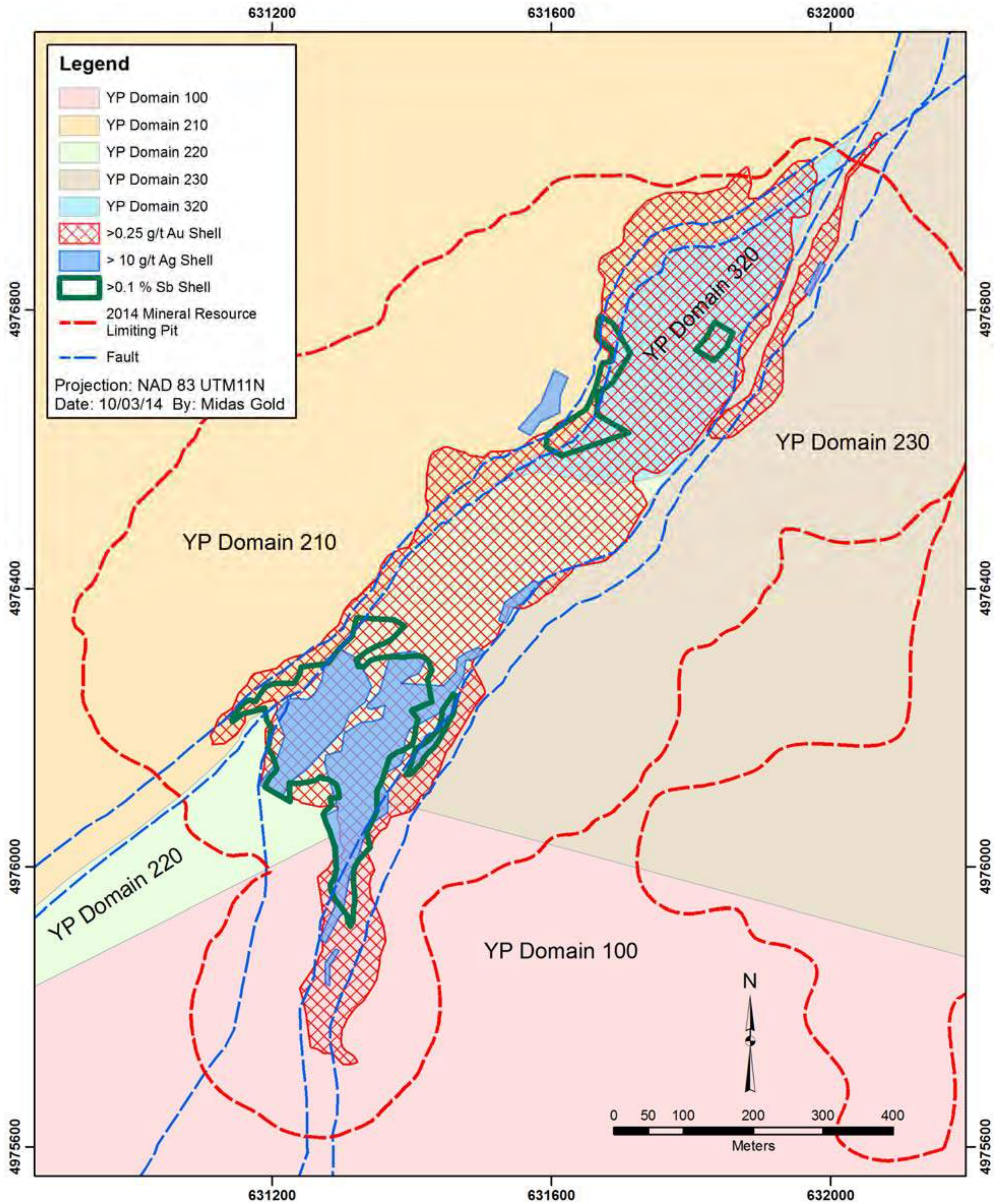


Table 14.20: Descriptive Statistics for Raw Assays for Yellow Pine

Metal	Au Fire Assay (g/t)						
Shell	Inside 0.25 g/t Au						Outside Au
Estimation Domain	All	100	210	220	230	320	All
Mean	1.725	0.976	1.130	2.154	0.872	1.560	0.076
Standard Error	0.014	0.037	0.030	0.019	0.029	0.037	0.002
Median	1.029	0.369	0.489	1.714	0.471	0.686	0.027
Standard Deviation	2.127	1.278	1.674	2.214	1.455	2.353	0.228
Minimum	0.003	0.003	0.003	0.003	0.003	0.003	0.003
Maximum	36.00	8.057	22.29	36.00	28.20	30.62	12.48
Count	23,868	1,208	3,199	13,007	2,498	3,956	16,012
CV	1.23	1.31	1.48	1.03	1.67	1.51	2.99
Metal	Ag (g/t)			Sb (%)			
Shell	10 g/t Ag	Inside Au	Outside Au	0.1 % Sb	Inside Au	Outside Au	
Mean	22.05	2.066	0.414	0.768	0.022	0.003	
Standard Error	1.083	0.038	0.010	0.025	0.001	0.000	
Median	10.97	1.030	0.250	0.290	0.003	0.001	
Standard Deviation	43.01	4.523	1.197	1.454	0.126	0.024	
Minimum	0.017	0.017	0.017	0.001	0.000	0.000	
Maximum	975.0	222.0	111.0	24.10	4.420	1.800	
Count	1,578	14,496	14,198	3,380	12,930	14,075	
CV	1.95	2.19	2.89	1.89	5.77	7.24	
Note: Assay information excludes un-assayed samples and samples within overburden.							

14.4.5 Compositing

Gold, antimony and silver were composited downhole on 3 m intervals within the gold grade shell. Composites at the end of drill holes that were <0.6 m in length were removed from the final data set. Prior to compositing, gold grade was checked relative to sample length and no correlation was found, indicating that capping of outliers can be applied to composites. Composites generated from missing assay data were removed from the data set discussed in the following sections and used for estimation. Descriptive statistics for raw composites are shown in Table 14.21.

Table 14.21: Yellow Pine Raw Composite Statistics by Estimation Domain

Statistic	All Domains	Domain 100	Domain 210	Domain 220	Domain 230	Domain 320
Uncapped Clustered Composites (in g/t Au) Outside 0.25 g/t Gold Shell (AuCode_990)						
Mean	0.073	0.069	0.079	0.059	0.074	0.083
Standard Error	0.002	0.005	0.004	0.002	0.003	0.006
Median	0.030	0.033	0.030	0.023	0.027	0.034
Standard Deviation	0.176	0.132	0.224	0.096	0.144	0.233
Minimum	0.003	0.003	0.003	0.003	0.003	0.003
Maximum	6.527	1.638	5.314	1.405	1.964	6.527
Count	9,518	590	2,567	2,026	2,915	1,420
Coefficient of Variation	2.41	1.90	2.83	1.63	1.95	2.81

Statistic	All Domains	Domain 100	Domain 210	Domain 220	Domain 230	Domain 320
Uncapped Clustered Composites (in g/t Au) Inside 0.25 g/t Gold Shell (AuCode_1000, 1100)						
Mean	1.687	1.001	1.102	2.125	0.860	1.578
Standard Error	0.016	0.046	0.034	0.023	0.032	0.045
Median	1.092	0.477	0.539	1.795	0.509	0.850
Standard Deviation	1.847	1.194	1.431	1.916	1.286	2.025
Minimum	0.003	0.003	0.003	0.003	0.003	0.003
Maximum	23.48	6.984	12.47	21.13	21.45	23.48
Count	13,087	670	1,796	6,967	1,589	2,065
Coefficient of Variation	1.10	1.19	1.30	0.90	1.49	1.28
Uncapped Clustered Composites (in g/t Ag) Outside 10 g/t Silver Shell (AgCode_1000)						
Mean	0.411	0.456	0.340	0.432	0.421	0.473
Standard Error	0.013	0.042	0.008	0.015	0.037	0.020
Median	0.250	0.250	0.250	0.250	0.250	0.250
Standard Deviation	1.196	0.890	0.358	0.645	1.935	0.688
Minimum	0.080	0.170	0.080	0.170	0.080	0.080
Maximum	89.96	11.27	6.609	7.499	89.96	7.799
Count	8,514	459	2,275	1,848	2,719	1,213
Coefficient of Variation	2.91	1.95	1.05	1.49	4.60	1.46
Uncapped Clustered Composites (in g/t Ag) Inside 10 g/t Silver Shell (AgCode_3000)						
Mean	21.93	22.81	46.51	19.56	42.57	
Standard Error	1.271	3.076	14.33	0.882	17.10	
Median	12.26	13.47	25.23	11.93	12.31	
Standard Deviation	37.08	24.99	79.80	23.52	110.8	
Minimum	0.170	0.382	3.262	0.170	0.170	
Maximum	712.6	146.8	348.4	197.3	712.6	
Count	851	66	31	712	42	0
Coefficient of Variation	1.69	1.10	1.72	1.20	2.60	
Uncapped Clustered Composites (in % Sb) Outside 0.1% Antimony Shell (SbCode_1000)						
Mean	0.019	0.044	0.019	0.025	0.007	0.005
Standard Error	0.001	0.006	0.003	0.002	0.001	0.000
Median	0.003	0.011	0.003	0.003	0.003	0.003
Standard Deviation	0.097	0.094	0.117	0.112	0.023	0.010
Minimum	0.000	0.001	0.000	0.000	0.000	0.000
Maximum	2.558	0.926	2.558	2.148	0.380	0.170
Count	7,249	269	1,285	3,744	1,358	593
Coefficient of Variation	5.00	2.14	6.19	4.59	3.03	1.98
Uncapped Clustered Composites (in % Sb) Inside 0.1% Antimony Shell (SbCode_2000)						
Mean	0.626	0.626	0.884	0.564	1.247	0.543
Standard Error	0.030	0.081	0.145	0.030	0.350	0.097
Median	0.267	0.254	0.376	0.264	0.253	0.349
Standard Deviation	1.099	0.985	1.770	0.912	2.098	0.563
Minimum	0.001	0.003	0.005	0.001	0.017	0.005
Maximum	16.96	5.442	16.96	11.72	8.462	2.310
Count	1,310	147	149	944	36	34
Coefficient of Variation	1.76	1.57	2.00	1.62	1.68	1.04

14.4.6 Evaluation of Outliers

To mitigate estimation risk associated with use of high-grade statistical outliers, capping grades were determined for each estimation domain based on inflection points on log-probability plots of the raw 3 m composites. Capping grades are shown in Table 14.22 as well as percentage of metal removed based on relative change in declustered average grade. Capped composite statistics are shown in Table 14.23. Low capping grades for antimony outside the antimony shell are warranted due to low average grade and small relative standard deviation. In general, the estimation domains and mineralized shells adequately subdivide samples into regions with distinct populations and coefficients of variation acceptably low for geostatistical estimation.

Table 14.22: Capping Grades for 3 m Composites

Statistic	Domain 100	Domain 210	Domain 220	Domain 230	Domain 320
Au (g/t) Inside 0.25 g/t Gold Shell (AuCode_1000, 1100)					
Ndat	670	1796	6967	1589	2064
Maximum Value	6.984	12.465	21.125	21.451	23.484
Cap Value	N/A	10	10	10	10
Number Capped	0	3	44	4	14
Mean Uncapped	0.845	0.944	1.403	0.860	1.301
Mean Capped	0.845	0.943	1.395	0.840	1.288
Lost Metal (%)	0	-0.1	-0.6	-2.3	-1
Ag (g/t) Inside 10 g/t Silver Shell (AgCode_3000)					
Ndat	66	31	712	42	0
Maximum Value	146.8	348.4	197.3	712.6	
Cap Value	80	50	120	100	
Number Capped	1	4	6	3	
Mean Uncapped	22.81	46.51	19.56	42.57	
Mean Capped	21.80	23.99	19.19	26.36	
Lost Metal (%)	-4.4	-48.4	-1.9	-38.1	
Ag (g/t) Outside 10 g/t Silver Shell (AgCode_1000)					
Ndat	382	1,454	3,722	1,360	1,296
Maximum Value	31.02	59.75	70.21	42.45	12.83
Cap Value	20	25	25	15	10
Number Capped	3	5	17	5	5
Mean Uncapped	1.937	1.850	2.478	1.539	1.517
Mean Capped	1.899	1.798	2.404	1.507	1.514
Lost Metal (%)	-1.9	-2.8	-3.0	-2.1	-0.2
Sb % Inside 0.1% Antimony Shell (SbCode_2000)					
Ndat	147	149	944	36	34
Maximum Value	5.442	16.962	11.72	8.462	2.31
Cap Value	N/A	6	9	5	1
Number Capped	0	2	1	4	4
Mean Uncapped	0.626	0.884	0.564	1.247	0.543
Mean Capped	0.626	0.788	0.561	1.087	0.445
Lost Metal (%)	0.0	-10.8	-0.5	-12.8	-18.1
Sb % Outside 0.1% Antimony Shell (SbCode_1000)					
Ndat	269	1,285	3,744	1,358	593
Maximum Value	0.926	2.558	2.148	0.38	0.17
Cap Value	N/A	0.5	0.5	0.3	N/A
Number Capped	0	11	40	2	0
Mean Uncapped	0.044	0.019	0.025	0.007	0.005
Mean Capped	0.044	0.014	0.020	0.007	0.005
Lost Metal (%)	0.0	-25.9	-18.0	-1.4	0.0

Table 14.23: Capped Clustered Composite Statistics for Yellow Pine

Statistic	All Domains	Domain 100	Domain 210	Domain 220	Domain 230	Domain 320
Capped Clustered Composites (in g/t Au) Outside 0.25 g/t Gold Shell (AuCode_990)						
Mean	0.073	0.069	0.079	0.059	0.074	0.083
Standard Error	0.002	0.005	0.004	0.002	0.003	0.006
Median	0.030	0.033	0.030	0.023	0.027	0.034
Standard Deviation	0.176	0.132	0.224	0.096	0.144	0.233
Minimum	0.003	0.003	0.003	0.003	0.003	0.003
Maximum	6.527	1.638	5.314	1.405	1.964	6.527
Count	9,518	590	2,567	2,026	2,915	1,420
Coefficient of Variation	2.41	1.90	2.83	1.63	1.95	2.81
Capped Clustered Composites (in g/t Au) Inside 0.25 g/t Gold Shell (AuCode_1000, 1100)						
Mean	1.671	1.001	1.098	2.107	0.840	1.558
Standard Error	0.015	0.046	0.033	0.022	0.028	0.042
Median	1.092	0.477	0.539	1.795	0.509	0.850
Standard Deviation	1.751	1.194	1.408	1.812	1.111	1.897
Minimum	0.003	0.003	0.003	0.003	0.003	0.003
Maximum	10.000	6.984	10.000	10.000	10.000	10.000
Count	13,087	670	1,796	6,967	1,589	2,065
Coefficient of Variation	1.05	1.19	1.28	0.86	1.32	1.22
Capped Clustered Composites (in g/t Ag) Outside 10 g/t Silver Shell (AgCode_1000)						
Mean	0.397	0.443	0.340	0.432	0.380	0.473
Standard Error	0.007	0.036	0.008	0.015	0.013	0.020
Median	0.250	0.250	0.250	0.250	0.250	0.250
Standard Deviation	0.607	0.761	0.358	0.645	0.669	0.688
Minimum	0.080	0.170	0.080	0.170	0.080	0.080
Maximum	10.000	7.000	6.610	7.500	10.000	7.800
Count	8,514	459	2,275	1,848	2,719	1,213
Coefficient of Variation	1.53	1.72	1.05	1.49	1.76	1.46
Capped Clustered Composites (in g/t Ag) Inside 10 g/t Silver Shell (AgCode_3000)						
Mean	19.92	21.80	23.99	19.19	26.36	
Standard Error	0.743	2.574	2.718	0.801	4.584	
Median	12.25	13.47	25.23	11.94	12.31	
Standard Deviation	21.67	20.91	15.14	21.38	29.71	
Minimum	0.170	0.380	3.260	0.170	0.170	
Maximum	120.0	80.00	50.00	120.0	100.0	
Count	851	66	31	712	42	0
Coefficient of Variation	1.09	0.96	0.63	1.11	1.13	

Statistic	All Domains	Domain 100	Domain 210	Domain 220	Domain 230	Domain 320
Capped Clustered Composites (in % Sb) Outside 0.1% Antimony Shell (SbCode_1000)						
Mean	0.017	0.043	0.016	0.022	0.006	0.005
Standard Error	0.001	0.006	0.002	0.002	0.001	0.000
Median	0.003	0.010	0.003	0.002	0.003	0.003
Standard Deviation	0.088	0.094	0.079	0.109	0.021	0.010
Minimum	0.000	0.001	0.000	0.000	0.000	0.000
Maximum	2.148	0.926	1.344	2.148	0.380	0.170
Count	7,249	269	1,285	3,744	1,358	593
Coefficient of Variation	5.15	2.18	4.99	5.06	3.48	1.98
Capped Clustered Composites (in % Sb) Inside 0.1% Antimony Shell (SbCode_2000)						
Mean	0.596	0.626	0.762	0.555	1.047	0.406
Standard Error	0.026	0.081	0.088	0.029	0.278	0.049
Median	0.265	0.254	0.376	0.262	0.253	0.349
Standard Deviation	0.943	0.985	1.079	0.883	1.670	0.288
Minimum	0.001	0.003	0.005	0.001	0.001	0.005
Maximum	9.000	5.442	6.000	9.000	5.000	1.000
Count	1,310	147	149	944	36	34
Coefficient of Variation	1.58	1.57	1.42	1.59	1.59	0.71

14.4.7 Statistical Analysis and Spatial Correlation

Exponential and spherical semi-variogram models were generated for gold, antimony and silver to guide the search ellipses and establish spatial correlation and sample weighting for the estimate. The nugget effect was derived using down-hole variograms. Semi-variogram models (Table 14.24) were based on experimental log variograms generated for each estimation domain within the grade shells using Snowden Supervisor™ software. Variography for gold and antimony indicates good spatial correlation along perceived structural trends such as major fault corridors and northeast striking splay structures to the MCFZ, with many directions of maximum continuity oriented sub-parallel to the intersection of northwest dipping and north-south striking faults.

Table 14.24: Semi-Variogram Models for Yellow Pine

Metal	Domain	Gemcom ZXZ Rotations(1)			Nugget C0	Sill C1 and C2	Ranges a1, a2 (m)			Model Structure Type
		Around Z	Around X	Around Z			SM X-ROT	MJ Y-ROT	MN Z-ROT	
Au	100	68	68	-106	0.05	0.17	4	4	14	Exp.
						0.94	69	50	22	Exp.
	210	62	-62	42	0.05	0.21	43	32	15	Spherical
						0.74	70	45	20	Spherical
	220	40	-85	-95	0.1	0.07	26	10	14	Exp.
						0.58	115	60	36	Exp.
	230	-128	75	-112	0.05	0.08	68	40	11	Exp.
						0.94	90	80	25	Exp.
	320	-135	53	-105	0.1	0.62	47	50	24	Exp.
						0.12	90	70	40	Exp.

Metal	Domain	Gemcom ZXZ Rotations(1)			Nugget C0	Sill C1 and C2	Ranges a1, a2 (m)			Model Structure Type
		Around Z	Around X	Around Z			SM X-ROT	MJ Y-ROT	MN Z-ROT	
Ag	100	82	83	-105	0.05	0.31	37	37	18	Exp.
						0.37	100	75	35	Exp.
	210	45	-59	8	0.06	0.37	96	10	12	Exp.
						0.49	120	80	25	Exp.
	220	70	-80	95	0.03	0.21	7	7	9	Exp.
						0.63	115	80	44	Exp.
	230	60	-82	15	0.07	0.45	18	18	9	Exp.
						0.71	105	80	19	Exp.
	320	31	-52	134	0.06	0.51	50	46	19	Exp.
						0.38	95	82	35	Exp.
Sb	100	75	-69	160	0.1	0.1	37	22	16	Exp.
						0.97	75	46	35	Exp.
	210	60	-75	67	0.05	0.26	9	9	9	Exp.
						0.48	95	65	25	Exp.
	220	60	-60	165	0.1	0.32	42	43	11	Exp.
						0.37	90	90	20	Exp.
	230	60	-75	135	0.03	0.35	43	30	12	Exp.
						0.42	80	55	22	Exp.
	320	30	-30	90	0.03	0.07	85	25	8	Spherical
						0.74	101	65	28	Spherical

Notes:
(1) Defined as positive rotation (right hand) of X around Z axis towards Y, rotation around newly created Z and Y axes around X, followed by rotation around newly created Z axis.

14.4.8 Block Model Parameters and Grade Estimation

The block model mineral resource estimate for Yellow Pine was developed with block dimensions of 12.19 x 12.19 x 6.096 m (40 x 40 x 20 ft) with coordinates defined in Table 14.25. Blocks were discretized into a 3 x 3 x 3 array of points during estimation.

Table 14.25: Block Model Definition for Yellow Pine

Deposit	Dimension (m)			Origin (m)			Number of Blocks			Rotation
	X	Y	Z	X	Y	Z	X	Y	Z	
Yellow Pine	12.192	12.192	6.096	630,686.096	4,795,346.096	2,295.152	151	161	152	0

Notes: Block centroid, NAD83 Zone 11N Datum

Density was estimated within grade shells in a single pass using inverse distance squared weighting with search ellipse orientations based on the gold variograms models. Un-estimated blocks were assigned the average density for the encompassing estimation domain or lithology solid.

The Yellow Pine mineral resource estimate was completed for gold, antimony and silver using the estimation domains and grade shells discussed above. Within the grade shells, blocks were estimated by ordinary Kriging using the capped 3 m composite file and the semi-variogram models discussed above. Grade shells were treated as hard boundaries and structural domains treated as soft boundaries for sample selection during estimation. Gold and antimony were estimated within and outside the grade shells, but only material within the shells is eligible for indicated classification. Table 14.26 summarizes the estimation parameters for Yellow Pine.

Table 14.26: Summary of Estimation Parameters for Yellow Pine

Au 1000 and 1100 Grade Shell																
Domain		100	100	100	210	210	210	220	220	220	230	230	230	320	320	320
Search Pass		1	2	3	1	2	3	1	2	3	1	2	3	1	2	3
Gemcom ZXZ Rotations	Around Z	68	68	68	62	62	62	40	40	40	-128	-128	-128	-135	-135	-135
	Around X	68	68	68	-62	-62	-62	-85	-85	-85	75	75	75	53	53	53
	Around Z	-106	-106	-106	42	42	42	-95	-95	-95	-112	-112	-112	-105	-105	-105
Search Ellipse Radius	x (m)	60	60	120	60	60	120	60	60	120	60	60	120	60	60	120
	y (m)	45	45	90	45	45	80	45	45	90	45	45	110	45	45	120
	z (m)	20	20	30	15	15	20	25	25	35	15	20	20	25	25	40
High Grade Search Limit	x (m)	45	45	45	45	45	45				45	45	45			
	y (m)	45	45	45	45	45	45				45	45	45			
	z (m)	20	20	20	15	15	15				15	15	15			
	Limit Value	5	5	5	10	10	10				10	10	10			
No of Samples	Min	4	6	4	4	6	4	4	6	4	4	6	4	4	6	4
	Max	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16
Max Samples	Per Hole	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
No of Holes Required		1	2	1	1	2	1	1	2	1	1	2	1	1	2	1
Search Type		Ellipsoidal	Octant	Ellipsoidal	Ellipsoidal	Octant	Ellipsoidal	Ellipsoidal	Octant	Ellipsoidal	Ellipsoidal	Octant	Ellipsoidal	Ellipsoidal	Octant	Ellipsoidal
Method		OK	OK	OK	OK	OK	OK	OK	OK	OK	OK	OK	OK	OK	OK	OK

Ag 3000 Grade Shell											
Domain		100	100	210	210	220	220	230	230	320	320
Search Pass		1	2	1	2	1	2	1	2	1	2
Gemcom ZXZ Rotations	Around Z	82	82	45	45	70	70	60	60	31	31
	Around X	83	83	-59	-59	-80	-80	-82	-82	-52	-52
	Around Z	-105	-105	8	8	95	95	15	15	20	20
Search Ellipse Radius	x (m)	60	120	60	120	60	120	60	120	60	120
	y (m)	45	90	45	80	45	80	45	90	45	100
	z (m)	20	30	15	20	20	40	15	20	20	30
High Grade Search Limit	x (m)					45	45				
	y (m)					45	45				
	z (m)					15	15				
	Limit Value					120.0	120.0				
No of Samples	Min	6	4	6	4	6	4	6	4	6	4
	Max	16	16	16	16	16	16	16	16	16	16
Max Samples	Per Hole	4	4	4	4	4	4	4	4	4	4
No of Holes Required		2	1	2	1	2	1	2	1	2	1
Search Type		Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal
Method		OK	OK	OK	OK	OK	OK	OK	OK	OK	OK

Ag Within 1000 Au Grade Shell											
Domain		100	100	210	210	220	220	230	230	320	320
Search Pass		1	2	1	2	1	2	1	2	1	2
Gemcom ZXZ Rotations	Around Z	82	82	45	45	70	70	60	60	31	31
	Around X	83	83	-59	-59	-80	-80	-82	-82	-52	-52
	Around Z	-105	-105	8	8	95	95	15	15	20	20
Search Ellipse Radius	x (m)	60	120	60	120	60	120	60	120	60	120
	y (m)	45	90	45	80	45	80	45	90	45	100
	z (m)	20	30	15	20	20	40	15	20	20	30
High Grade Search Limit	x (m)					45	45				
	y (m)					45	45				
	z (m)					15	15				
	Limit Value					120.0	120.0				
No of Samples	Min	6	4	6	4	6	4	6	4	6	4
	Max	16	16	16	16	16	16	16	16	16	16
Max Samples	Per Hole	4	4	4	4	4	4	4	4	4	4
No of Holes Required		2	1	2	1	2	1	2	1	2	1
Search Type		Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal
Method		OK	OK	OK	OK	OK	OK	OK	OK	OK	OK

Sb 2000 Grade Shell											
Domain		100	100	210	210	220	220	230	230	320	320
Search Pass		1	2	1	2	1	2	1	2	1	2
Gemcom ZXZ Rotations	Around Z	75	75	60	60	60	60	60	60	30	30
	Around X	-69	-69	-75	-75	-60	-60	-75	-75	-30	-30
	Around Z	160	160	67	67	165	165	135	135	90	90
Search Ellipse Radius	x (m)	60	120	60	120	60	120	60	120	60	120
	y (m)	45	80	45	80	45	90	45	80	45	80
	z (m)	20	30	15	20	20	30	15	20	20	30
High Grade Search Limit	x (m)			45	45						
	y (m)			45	45						
	z (m)			15	15						
	Limit Value			6	6						
No of Samples	Min	6	4	6	4	6	4	6	4	6	4
	Max	16	16	16	16	16	16	16	16	16	16
Max Samples	Per Hole	4	4	4	4	4	4	4	4	4	4
No of Holes Required		2	1	2	1	2	1	2	1	2	1
Search Type		Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal
Method		OK	OK	OK	OK	OK	OK	OK	OK	OK	OK

Sb Within 1000 Au Grade Shell											
Domain		100	100	210	210	220	220	230	230	320	320
Search Pass		1	2	1	2	1	2	1	2	1	2
Gemcom ZXZ Rotations	Around Z	75	75	60	60	60	60	60	60	30	30
	Around X	-69	-69	-75	-75	-60	-60	-75	-75	-30	-30
	Around Z	160	160	67	67	165	165	135	135	90	90
Search Ellipse Radius	x (m)	60	120	60	120	60	120	60	120	60	120
	y (m)	45	80	45	80	45	90	45	80	45	80
	z (m)	20	30	15	20	20	30	15	20	20	30
High Grade Search Limit	x (m)	25	25	25	25	25	25	25	25	25	25
	y (m)	25	25	25	25	25	25	25	25	25	25
	z (m)	10	10	10	10	10	10	10	10	10	10
	Limit Value	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
No of Samples	Min	6	4	6	4	6	4	6	4	6	4
	Max	16	16	16	16	16	16	16	16	16	16
Max Samples	Per Hole	4	4	4	4	4	4	4	4	4	4
No of Holes Required		2	1	2	1	2	1	2	1	2	1
Search Type		Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal
Method		OK	OK	OK	OK	OK	OK	OK	OK	OK	OK

		SG Ore (Within 1000 Au Grade Shell)					SG Waste (Outside the Au Grade Shell)				
Domain		100	210	220	230	320	100	210	220	230	320
Search Pass		1	2	3	2	3	1	2	3	2	3
Gemcom ZXZ Rotations	Around Z	68	62	40	-128	-135	68	62	40	-128	-135
	Around X	68	-62	-85	75	53	68	-62	-85	75	53
	Around Z	-106	42	-95	-112	-105	-106	42	-95	-112	-105
Search Ellipse Radius	x (m)	120	120	120	120	120	120	120	120	120	120
	y (m)	120	120	120	120	120	120	120	120	120	120
	z (m)	50	50	50	50	50	50	50	50	50	50
No of Samples	Min	2	2	2	2	2	2	2	2	2	2
	Max	5	5	5	5	5	5	5	5	5	5
Max Samples	Per Hole										
No of Holes Required		1	1	1	1	1	1	1	1	1	1
Search Type		Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal
Method		ID2	ID2	ID2	ID2	ID2	ID2	ID2	ID2	ID2	ID2

The gold estimation strategy was designed to limit the influence of historical underground holes drilled by Bradley, which show an apparent high bias presumably associated with their location in the highest-grade portions of the deposit, as discussed in Section 12. The first pass only estimated blocks within 12 m of Bradley underground drill holes using all composites. In the second pass, search ellipsoid radii were adjusted to represent half the variograms continuity and composite samples from the underground Bradley drill holes were removed from the sample set. The third pass was adjusted as needed to estimate any remaining blocks. Antimony was estimated with two passes of which the first pass search ellipsoid radii represent approximately half of the variogram range of continuity. The second pass was designed to estimate any remaining blocks. Only composite samples from the Midas Gold, Barrick Gold Corporation (**Barrick**), Ranchers and United States Bureau of Mines (**USBM**) drilling campaigns were used to estimate antimony; Bradley samples were excluded due to apparent high bias with respect to antimony grade, even though some of these holes were focused within the highest grades portions of the antimony mineralization.

14.4.9 Block Model Validation

The block model for Yellow Pine was validated by completing a series of graphical inspections, bias checks, sensitivity studies and comparison to prior estimates. Graphically, the model was validated by visually comparing the composites to estimated block grades on plan and section views. Local bias was assessed by comparing the average composite grade against the encompassing block for both gold and antimony and by comparison of the average declustered composite grade and nearest-neighbor estimate to the Kriged estimate on swath plots in the X, Y and Z directions (Figure 14.13, Figure 14.14, and Figure 14.15, respectively). The resultant histograms for gold composites vs. block grades compare well and indicate that the block values are similar to the composite datasets. The histograms for antimony have increased kurtosis indicating a degree of smoothing in the antimony estimate. Model sensitivities were run to assess the impact of historical data on the estimate. Exclusion of the pre-1953 drill hole data results in a 2.4% reduction in average gold grade and an approximate 4% reduction in contained gold at a 0.75 g/t Au cutoff grade, reported within a conceptual pit shell.

Figure 14.13: East-West Validation Gold Swath Plot for Yellow Pine

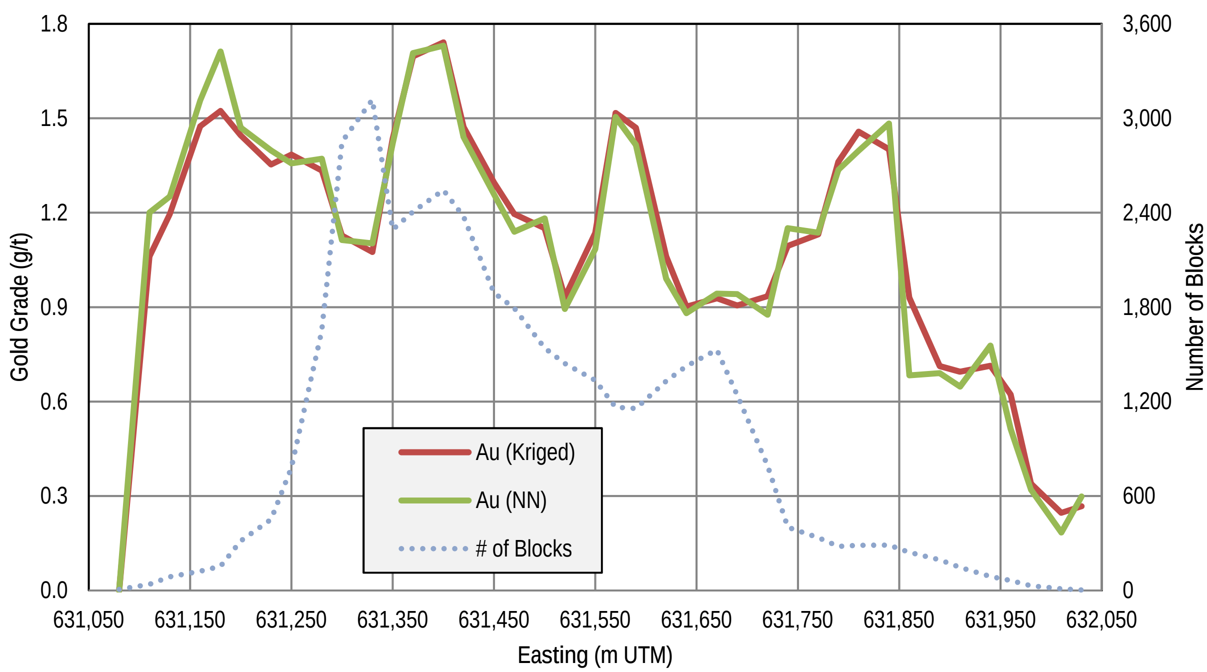


Figure 14.14: North-South Gold Validation Swath Plot for Yellow Pine

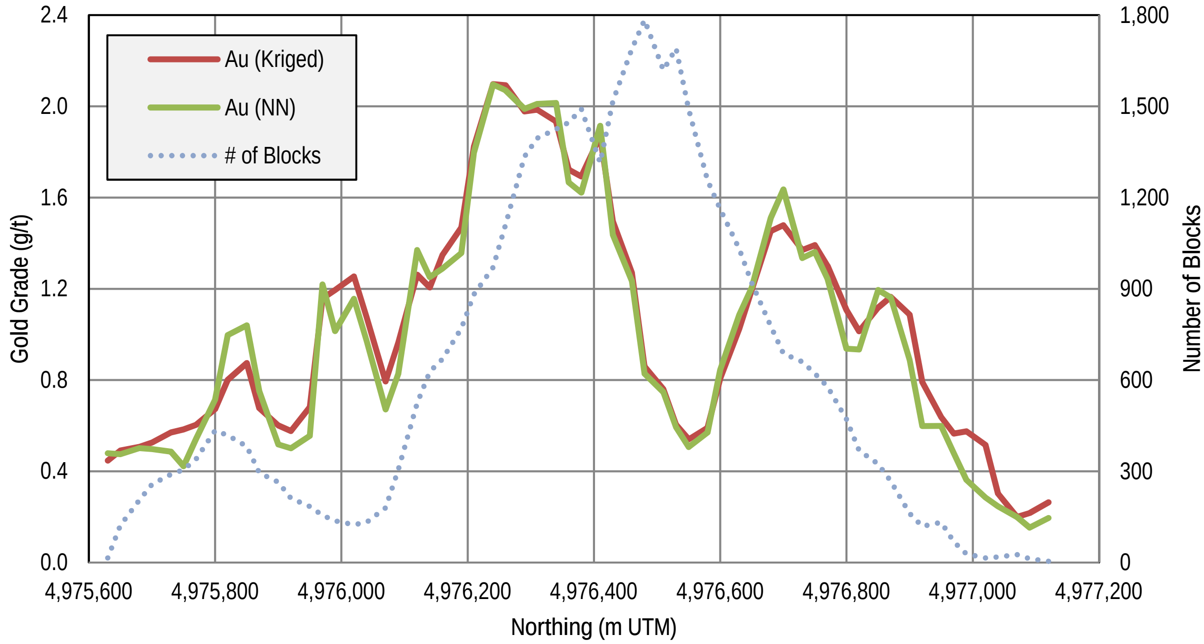
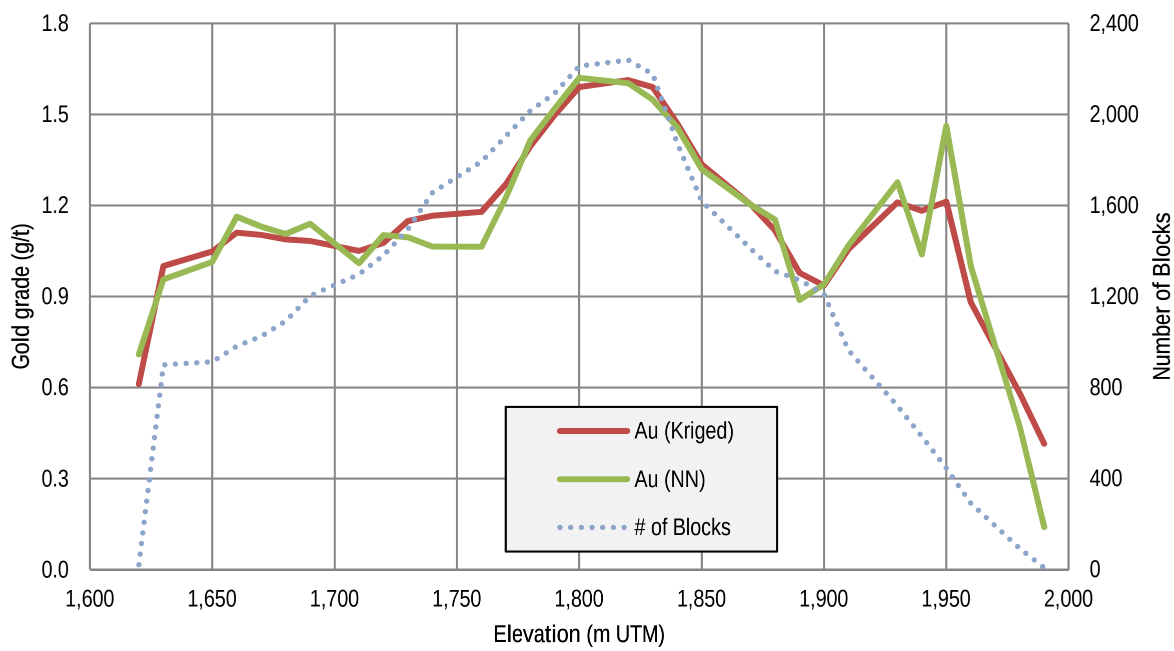


Figure 14.15: Elevation Validation Gold Swath Plot for Yellow Pine



14.4.10 Yellow Pine Mineral Resource Classification

Confidence criteria used to guide mineral resource classification at Yellow Pine included search and composite selection, spatial distribution of samples and well-demonstrated support from "modern" drill data so that grades are not significantly biased by pre-1953 drill holes. This latter requirement ensures that the influence of historic data is partially diminished by the presence of "modern" data in the same area; "modern" data is defined as composites from the post-1980 Midas Gold, Ranchers and Barrick drilling campaigns. Blocks eligible for indicated classification were restricted to those within the gold grade shell and flagged using a separate classification search pass utilizing a 45 x 35 x 25 m search ellipse representing approximately 85% of the modeled variogram sill and requiring at least four samples from two drill holes occurring in at least three octants from "modern" drill data. Final classification was applied following manual smoothing of the results to encompass zones predominantly flagged as indicated. The antimony mineral resource estimates were not classified separately and are instead reported with the gold classification categories.

14.5 HISTORIC TAILINGS

14.5.1 Mineral Resource Estimation Procedures

The Historic Tailings mineral resource estimate is based on the current drill hole database, geologic model of tailings, and LiDAR topographic data. The geologic modeling and estimation of mineral resources was completed using the commercial three-dimensional block modelling and mine planning software packages Geovia GEMS™ 6.6 and Micromine™ version 14; geostatistical analysis was completed using Isaaks & Co.'s SAGE2001™ software package.

14.5.2 Drill Hole Database

The drill hole database, supplied by Midas Gold as an Excel Workbook, contained collar locations surveyed in UTM grid coordinates, assay intervals with gold, antimony, and silver analyses by fire assay and/or cyanide soluble assay, geologic intervals with rock types and in situ density measurements. The database contained data for 73 separate drill holes representing a mixture of historic and modern drilling programs. Some drill holes were not assayed and only used for the establishment the upper and lower boundaries of the tailings and some drill holes did not intercept tailings material. Only Midas Gold drill holes were used in the mineral resource estimate and primarily consist of hollow-stem auger drill holes completed in 2013 with some sonic drill holes completed in 2012. Samples not intersecting tailings material were removed from the data set utilized for estimation.

For the Historic Tailings deposit, antimony and silver mineral resources were calculated in addition to gold. Table 14.27 illustrates that the metal values for gold, antimony, and silver in Midas Gold drill holes were consistently analyzed for all sample intervals throughout the dataset utilized for estimation.

Table 14.27: Drill Hole Data used in the Historic Tailings Mineral Resource Estimate

Element	# Holes	# Assays	Meters
Gold	41	540	339
Antimony	41	540	339
Silver	41	540	339

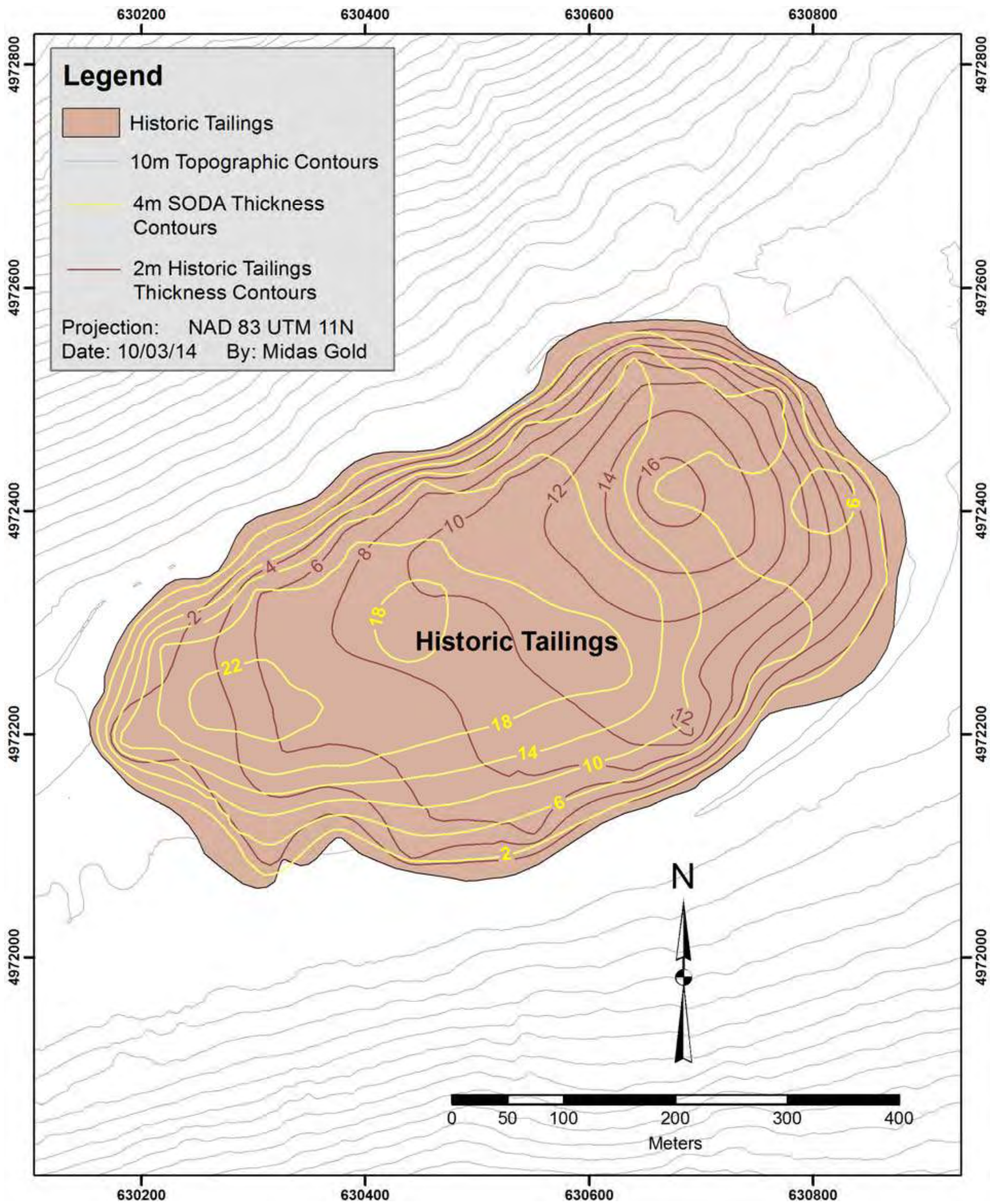
Assays lengths for the auger drill holes were typically 0.61 m (2 ft) which comprises the bulk of the data set. Sonic drill holes present in the deposit were typically assayed on 3.05 m (10 ft) intervals. All drill holes are vertical and the average drill hole spacing is approximately 60 m oriented along a grid rotated to an azimuth of 23 degrees.

14.5.3 Geologic Modeling

The Historic Tailings were hydraulically deposited within the Meadow Creek Valley from the 1920s through the 1950s; the tailings were generated from the Bradley sulfide flotation milling operations. The tailings were later overlain by spent heap leach ore from the 1980s through the 1990s heap leach operations; this area is often referred to as the Spent Ore Disposal Area (SODA). The Historic Tailings deposit is up to 18 m thick, with an average thickness of 6 m; the overlying spent ore material is up to 23 m thick, with an average thickness of 11 m. Historic

Tailings material was wire-framed based on drill hole intercepts, modern LiDAR and orthographic photos, and historic engineering drawings and airborne photos. The total volume of the Historic Tailings wireframe is 1,925,923 m^3.

Figure 14.16: Isopach of the Historic Tailings Deposit



14.5.4 Estimation Domain Modeling

The Historic Tailings solid serves as the only estimation domain utilized in the mineral resource estimate. Descriptive statistics for raw assays within the tailings solid are presented in Table 14.28. The drill holes were drilled on a quasi-regular grid and there is no evidence of data clustering in high- or low-grade areas. Higher-grade material shows northwest trends which shift location vertically, consistent with presumed deposition in tailings beaches during historical operations.

Table 14.28: Raw Assay Statistics for the Historic Tailings

Statistic	Width (m)	Au	Sb	Ag
Count	540	540	540	540
Mean	0.63	1.191	0.173	3.08
Standard Deviation	0.27	0.527	0.117	2.70
Range	2.90	4.636	0.996	41.75
Minimum	0.15	0.054	0.0045	0.25
Lower Quartile	0.61	0.813	0.090	1.80
Median	0.61	1.088	0.169	2.80
Upper Quartile	0.61	1.395	0.237	3.80
Maximum	3.05	4.690	1.000	42.00
Coefficient of Variation	0.42	0.44	0.68	0.88
95% Percentile	0.61	2.211	0.360	5.81
98% Percentile	1.52	2.704	0.440	7.04
99% Percentile	1.83	2.960	0.473	7.50

14.5.5 Compositing

Samples were composited on intervals of 0.61 m and 1.52 m, of which the 0.61 m composites were determined to exhibit a more regular distribution of composite lengths and were selected for estimation. Gold, antimony and silver were composited downhole within the Historic Tailings solid. The composited data yields a histogram with a moderately skewed distribution and very few samples with a grade <0.6 g/t Au.

14.5.6 Evaluation of Outliers

To evaluate potential risk associated with use of high-grade statistical outliers, potential capping grades were assessed using log-probability plots and by analysis of contained metal in deciles and centiles, following the Parrish Method (Parrish, 1997). These results indicate no cap for gold, a cap of 0.6% for antimony and 10 g/t for silver. Descriptive statistics for capped composites are presented in Table 14.29. Note that compositing of the longer sample intervals to 0.61 m yielded more composites than raw assays.

Table 14.29: Historic Tailings Descriptive Statistics for Capped Composites

Statistic	Au (g/t)(1)	Sb_Cap (%)(1)	Ag_Cap (g/t)(1)
Count(1)	568	568	568
Mean	1.183	0.173	2.98
Standard Deviation	0.488	0.104	1.52
Minimum	0.34	0.0045	0.5
Median	1.089	0.167	2.8
Maximum	4.69	0.6	10
Coefficient of Variation	0.412	0.598	0.51
95% Percentile	2.127	0.34	5.8
98% Percentile	2.392	0.414	6.81
99% Percentile	2.701	0.458	7.33

Note:
(1) Units apply to all statistics except "Count".

14.5.7 Statistical Analysis and Spatial Correlation

Correlogram models were developed using the SAGE2001™ software package to guide the search ellipse and establish spatial correlation and sample weighting for the estimate. Correlograms demonstrate low nugget effect with effective ranges close to the drill hole spacing and confirm the northwest anisotropy and stratified nature of the deposit. The correlogram is summarized in Table 14.30.

Table 14.30: Correlogram Models for the Historic Tailings

Metal	Ellipse Axes Azimuth/Plunge(1)			Nugget C0	Sill C1	Ranges a1 (m)			Type
	1st	2nd	3rd			1st	2nd	3rd	
Au	327/2	57/-2	106/87	0.028	0.972	82	50	3	Exp
Sb	336/1	66/-5	76/85	0.02	0.98	133	41	5	Exp
Ag	131/1	41/7	228/83	0.001	0.999	142	63	6	Exp

Note:
(1) Negative plunge is downward.

14.5.8 Block Model Parameters and Grade Estimation

Due to the unconsolidated and stratiform nature of the tailings material, the block model is defined assuming selective mining methods with excellent grade control. Block dimensions are 15.24 x 15.24 x 1.524 m (50 x 50 x 5 ft) with location summarized in Table 14.31. Blocks located partially within the solid were assigned a percent value for reporting purposes.

Table 14.31: Historic Tailings Block Model Definition

Deposit	Dimension (m)			Origin (m)			Number of Blocks			Rotation
	X	Y	Z	X	Y	Z	X	Y	Z	
Historic Tailings	15.24	15.24	1.524	630,007	4,972,007	1,982	68	48	40	0

Notes: Block centroid, NAD83 Zone 11N Datum

The Historic Tailings mineral resource was estimated using ordinary Kriging in a single pass using a search ellipse and sample weighting established by the correlogram models discussed above. Samples were limited to a maximum of 3 per drill hole, increasing the influence of samples from neighboring drill holes. Search ellipse and sample selection parameters are summarized in Table 14.32. Density was estimated from 35 Shelby tube samples of the tailings material; the average dry density of the deposit was calculated to be 1.504 g/cm^3.

Table 14.32: Summary of Estimation Parameters for the Historic Tailings

Description	Au	Sb	Ag
Method	OK	OK	OK
Principal Axis Azimuth / Plunge	325/0	330/0	310/0
Intermediate Axis Azimuth / Plunge	055/0	60/0	40/0
Minor Axis Azimuth / Plunge	0/-90	0/-90	0/-90
Principle Axis Search Distance (m)	175	220	200
Major / Intermediate / Minor Axis	1 / 0.66 / 0.1	1 / 0.5 / 0.1	1 / 0.5 / 0.1
Search Type	Open	Open	Open
Composite Restrictions	Hard	Hard	Hard
Maximum Composites / Sector	N/A	N/A	N/A
Minimum Composites	1	2	3
Minimum # Holes	N/A	N/A	N/A
Maximum Composites / Hole	3	3	3
Maximum Composites	12	12	12

14.5.9 Block Model Validation

The block model for the Historic Tailings was validated by completing a series of graphical inspections, bias checks and reconciliation with historic production records. The block estimates and block percentages were reviewed visually relative to the composite grades and the tailings wireframe. Global bias was assessed by comparison of the Kriged estimate to the nearest neighbor estimate and showed a 3.5% variance. Local bias was assessed by way of a swath plot in the Z direction. Relative to the nearest neighbor estimate, the Kriged estimate displays local low bias on upper level benches, proximal to very high-grade composites in three different drill holes. The results of the Kriged model are generally consistent with estimates of metals reporting to the tailings calculated as the difference between historical mill-feed grade and recovered metal from historical Bradley Mining Company production records.

14.5.10 Historic Tailings Mineral Resource Classification

Confidence criteria used to guide the mineral resource classification includes Kriging variance and anisotropic minimum distance to the nearest composite. Final classification was assigned by digitizing contours around blocks with Kriging variance >0.66 and minimum distance >60 m for the gold estimate, to define areas of inferred classification. The inferred blocks are primarily located on the southern and western margins of the tailings solid where drill data is sparse.

14.6 ECONOMIC CRITERIA AND PIT OPTIMIZATIONS

CIM defines mineral resources as having “reasonable prospects for eventual economic extraction” requiring that mineralization meet certain grade and material volume thresholds sufficient for eventual economic extraction under reasonable production and recovery scenarios at reasonable cutoff grades. Prospects for eventual economic extraction were assessed using an open-pit optimization Lerchs-Grossman algorithm in MineSight® Version 9.00 software. Input parameters were developed from preliminary cost estimates and metallurgical recoveries from preliminary engineering studies, as show in Table 14.33.

Table 14.33: Pit Optimization Parameters by Deposit

Input Parameters	Units	Yellow Pine	Hangar Flats	West End	Historic Tailings	Notes
Mining Cost – Mineral Resource	$/t mined	1.75	1.75	1.75	1.75	*Includes mining G&A*
Mining Cost - Waste	$/t mined	1.90	1.50	1.75	0.50	*Includes mining G&A*
Oxide Processing Cost	$/t mined	N/A	N/A	9.00	N/A	*Excludes G&A costs*
Oxide Au Recovery	%	N/A	N/A	84 * AuCN/ AuFA+8.52	N/A	*Formula based on PFS level metallurgical test results*
Oxide / Sulfide Boundary	CN Au : FA Au	N/A	N/A	0.70	N/A	
Sulfide Processing Cost	$/t milled	16.50	16.50	16.50	16.50	*Excludes G&A costs*
Sulfide Au Recovery	%	93.0	92.0	88.0	80.0	
Dore Transport Cost	$/oz Au	1.15	1.15	1.15	1.15	
Dore Refining Cost	$/oz Au	1.00	1.00	1.00	1.00	
G & A + Rehabilitation Cost	$/t milled	3.50	3.50	3.50	3.50	
Pit Slopes	degrees	48	48	48	N/A	
Au Payability	%	99.5	99.5	99.5	99.5	
Au Selling Price - Initial Case(1)	$/oz	1,400	1,400	1,400	1,400	
Mining Dilution	%	0	0	0	0	
Mining Recovery	%	100	100	100	100	

Note:
(1) See Section 14.7 for comments on effective metal price used; while the mineral resource estimates were estimated using a $1,400/oz gold price, they are reported at higher cutoffs than these parameters generated, which equates to assuming a lower gold price.

Assumptions used to derive the cutoff grades and define the resource-limiting pits were estimated in order to meet the NI43-101 requirement for mineral resource estimates to demonstrate "reasonable prospects for eventual economic extraction" and vary from those used to limit the mineral reserves reported herein.

Because of the flat and shallow geometry of the Historic Tailings deposit, and due to potential use of the overlying material in conceptual construction scenarios, economic criteria were not assessed using a pit optimization. Instead, cost estimates for removing the overlying SODA material were compared to potential revenue from processing the tailings material and were shown to be positive.

14.7 MINERAL RESOURCE STATEMENTS

Mineral resources presented herein comply with guidelines of the Canadian Securities Administrators' National Instrument 43-101 and conform to CIM Definitions and Standards for Mineral Resources and Mineral Reserves (CIM, 2014). The mineral resources reported in Table 14.34 to Table 14.39, inclusively, are contained entirely within conceptual pit shells developed from the parameters discussed above. Based on these parameters, cutoff grades for Hangar Flats, West End and Yellow Pine were calculated at based on a $1,400/oz gold selling price, which resulted in an open pit sulfide cutoff grade, excluding adjustments, of approximately 0.55 g/t Au and an open pit oxide cutoff grade, excluding adjustments, of approximately 0.35 g/t Au. However, Midas Gold elected to report its base case mineral resource estimate using a 0.75 g/t Au sulfide cutoff grade and 0.45 g/t Au oxide cutoff grade which is equivalent to utilizing the cost assumptions stated in Table 14.33 and a gold selling price of approximately $1,000/oz for sulfide material and $1,100/oz for oxide material. Only mineral resources above these cutoffs and within the mineral resource-limiting pits are reported and, as such, mineralization falling below this cutoff grade or outside the mineral resource-limiting pit is not reported, irrespective of the grade. Sensitivity to cutoff grade is reported in Table 14.40, Figure 14.17, Figure 14.18, Figure 14.19, Figure 14.20, and Figure 14.21.

The Yellow Pine and Hangar Flats deposits contain zones with substantially elevated antimony-silver mineralization, defined as containing greater than 0.1% antimony, relative to the overall mineral resource. The existing Historic Tailings mineral resource also contains elevated concentrations of antimony. These higher-grade antimony zones are reported separately in Table 14.34 below to illustrate the potential for antimony production from the Project and are contained within the overall mineral resource estimates reported herein. Antimony zones are reported only if they lie within gold mineral resource estimates.

Table 14.34: Consolidated Mineral Resource Statement for the Stibnite Gold Project

Classification	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated							
Hangar Flats	21,389	1.60	1,103	4.30	2,960	0.11	54,180
West End	35,974	1.30	1,501	1.35	1,567	0.008	6,563
Yellow Pine	44,559	1.93	2,762	2.89	4,133	0.09	84,777
Historic Tailings	2,583	1.19	99	2.95	245	0.17	9,648
Total Indicated	*104,506*	*1.63*	*5,464*	*2.65*	*8,904*	*0.07*	*155,169*
Inferred							
Hangar Flats	7,451	1.52	363	4.61	1,105	0.11	18,727
West End	8,546	1.15	317	0.68	187	0.006	1,083
Yellow Pine	9,031	1.31	380	1.50	437	0.03	5,535
Historic Tailings	140	1.23	6	2.88	13	0.18	563
Total Inferred	*25,168*	*1.32*	*1,066*	*2.15*	*1,743*	*0.05*	*25,908*

Notes:

(1) All Mineral Resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum ("CIM") definitions, as required under National Instrument 43-101 ("NI43-101").

*(2) Mineral Resources are reported in relation to a conceptual pit shell in order to demonstrate potential for economic viability, as required under NI43-101; mineralization lying outside of these pit shells is not reported as a Mineral Resource. **Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. These Mineral Resource estimates include Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated.** All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely.*

(3) Open pit sulfide Mineral Resources are reported at a cutoff grade of 0.75 g/t Au and open pit oxide Mineral Resources are reported at a cutoff grade of 0.45 g/t Au.

Table 14.35: Antimony Sub-Domains Consolidated Mineral Resource Statement

Classification	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated							
Hangar Flats	3,901	2.06	258	7.23	907	0.59	50,729
Yellow Pine	6,080	2.27	443	6.99	1,367	0.58	77,841
Historic Tailings	2,583	1.19	99	2.95	245	0.17	9,648
Total Indicated	*12,564*	*1.98*	*800*	*6.23*	*2,518*	*0.50*	*138,218*
Inferred							
Hangar Flats	1,186	1.94	74	8.05	307	0.68	17,844
Yellow Pine	409	1.36	18	4.86	64	0.50	4,552
Historic Tailings	140	1.23	6	2.88	13	0.18	563
Total Inferred	*1,735*	*1.74*	*97*	*6.88*	*384*	*0.60*	*22,959*

Notes:
(1) Antimony mineral resources are reported as a subset of the total mineral resource within the conceptual pit shells used to constrain the total mineral resource in order to demonstrate potential for economic viability, as required under NI43-101; mineralization outside of these pit shells is not reported as a mineral resource. ***Mineral resources are not mineral reserves and do not have demonstrated economic viability.*** *All figures are rounded to reflect the relative accuracy of the estimate.*
(2) Open pit antimony sulfide mineral resources are reported at a cutoff grade 0.1% antimony within the overall 0.75 g/t Au cutoff.

Table 14.36: Hangar Flats Mineral Resource Statement Open Pit Sulfide at a 0.75 g/t Au Cutoff

Classification	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated	21,389	1.60	1,103	4.30	2,960	0.11	54,180
Inferred	7,451	1.52	363	4.61	1,105	0.11	18,727

Notes:
(1) Mineral resources are reported in relation to a conceptual pit shell in order to demonstrate potential for economic viability, as required under NI43-101; mineralization lying outside of these pit shells is not reported as a mineral resource. ***Mineral resources are not mineral reserves and do not have demonstrated economic viability. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated.*** *All figures are rounded to reflect the relative accuracy of the estimate.*
(2) Open pit sulfide mineral resources are reported at a cutoff grade of 0.75 g/t Au. The mineral resources were estimated based on the open pit optimization parameters listed in Table 14.33 and a gold selling price of US$1,400/oz, which resulted in an open pit sulfide cutoff grade of approximately 0.55 g/t Au. The 0.75 g/t Au sulfide cutoff grade is equivalent to utilizing the cost assumptions stated in Table 14.33 and a gold selling price of approximately $1,000/oz.

Table 14.37: West End Mineral Resource Statement Open Pit Oxide + Sulfide

Classification and Material Type	Cutoff Grade (g/t Au)	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated Oxide	0.45	8,448	0.80	216	1.22	332	0.010	1,769
Indicated Sulfide	0.75	27,526	1.45	1,285	1.40	1,235	0.008	4,794
Total Indicated		*35,974*	*1.30*	*1,501*	*1.35*	*1,567*	*0.008*	*6,563*
Inferred Oxide	0.45	2,057	0.76	50	0.40	27	0.004	168
Inferred Sulfide	0.75	6,489	1.28	267	0.77	161	0.006	916
Total Inferred		*8,546*	*1.15*	*317*	*0.68*	*187*	*0.006*	*1,083*

Notes:
(1) *Mineral resources are reported in relation to a conceptual pit shell in order to demonstrate potential for economic viability, as required under NI43-101; mineralization lying outside of these pit shells is not reported as a mineral resource.* ***Mineral resources are not mineral reserves and do not have demonstrated economic viability. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated.*** *All figures are rounded to reflect the relative accuracy of the estimate.*
(2) *Open pit sulfide mineral resources are reported at a cutoff grade of 0.75 g/t Au and open pit oxide mineral resources are reported at a cutoff grade of 0.45 g/t Au. The mineral resources were estimated based on the open pit optimization parameters listed in Table 14.33 and a gold selling price of US$1,400/oz, which resulted in an open pit sulfide cutoff grade of approximately 0.55 g/t Au and an open pit oxide cutoff grade of approximately 0.35 g/t Au. The 0.75 g/t Au sulfide cutoff grade and 0.45 g/t Au oxide cutoff grade are equivalent to utilizing the cost assumptions stated in Table 14.33 and a gold selling price of approximately $1,000/oz for sulfides and $1,100/oz for oxides.*

Table 14.38: Yellow Pine Mineral Resource Statement Open Pit Sulfide at a 0.75 g/t Au Cutoff

Classification	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated	44,559	1.93	2,762	2.89	4,133	0.09	84,777
Inferred	9,031	1.31	380	1.50	437	0.03	5,535

Notes:
(1) *Mineral resources are reported in relation to a conceptual pit shell in order to demonstrate potential for economic viability, as required under NI43-101; mineralization lying outside of these pit shells is not reported as a mineral resource.* ***Mineral resources are not mineral reserves and do not have demonstrated economic viability. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated.*** *All figures are rounded to reflect the relative accuracy of the estimate.*
(2) *Open pit sulfide mineral resources are reported at a cutoff grade of 0.75 g/t Au. The mineral resources were estimated based on the open pit optimization parameters listed in Table 14.33 and a gold selling price of US$1,400/oz, which resulted in an open pit sulfide cutoff grade of approximately 0.55 g/t Au. The 0.75 g/t Au sulfide cutoff grade is equivalent to utilizing the cost assumptions stated in Table 14.33 and a gold selling price of approximately $1,000/oz.*

Table 14.39: Historic Tailings Mineral Resource Statement Open Pit Sulfide at a 0.75 g/t Au Cutoff

Classification	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated	2,583	1.19	99	2.95	245	0.17	9,648
Inferred	140	1.23	6	2.88	13	0.18	563

Notes:
(1) *Mineral resources are reported in total above cutoff since all the spent heap leach ore stacked on top of the tailings would be removed for construction purposes and the tailings full exposed. Mineral resources are not mineral reserves and do not have demonstrated economic viability. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated. All figures are rounded to reflect the relative accuracy of the estimate.*
(2) *Open pit sulfide mineral resources are reported at a cutoff grade of 0.75 g/t Au. The mineral resources were estimated based on the open pit optimization parameters listed in Table 14.33 and a gold selling price of US$1,400/oz, which resulted in an open pit sulfide cutoff grade of approximately 0.55 g/t Au. The 0.75 g/t Au sulfide cutoff grade is equivalent to utilizing the cost assumptions stated in Table 14.33 and a gold selling price of approximately $1,000/oz.*

14.8 GRADE SENSITIVITY ANALYSIS

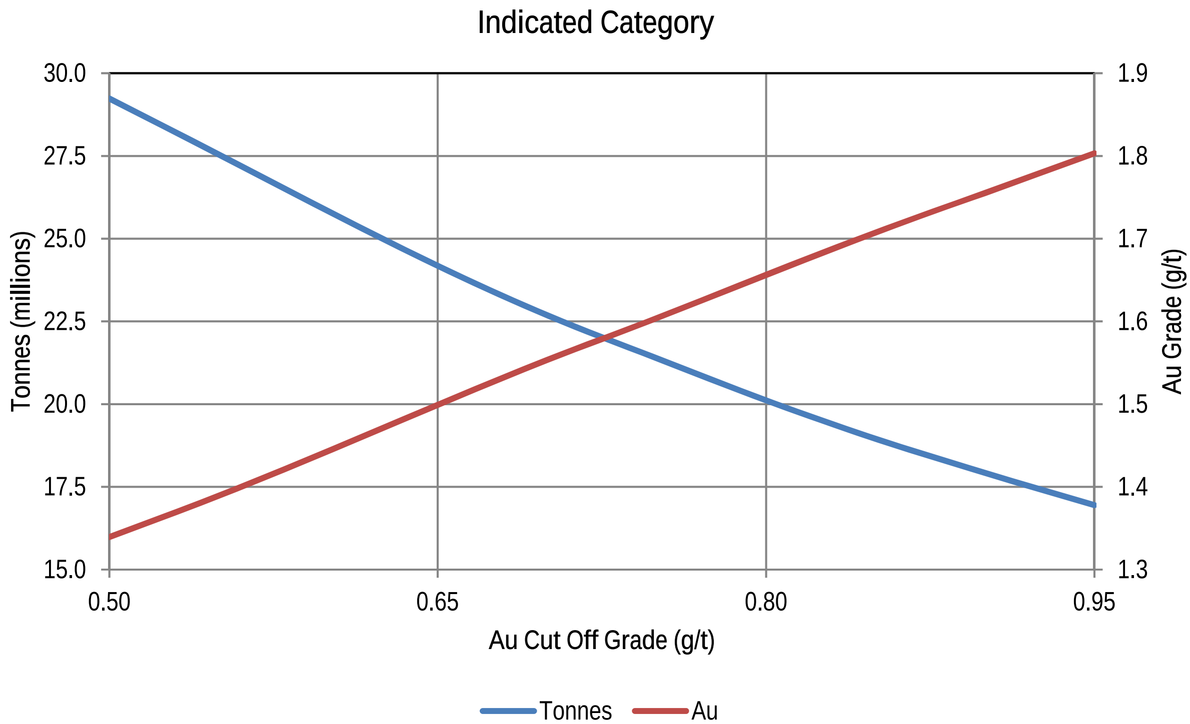
The mineral resources are sensitive to the gold cutoff grade used for reporting. To demonstrate this, the mineral resources are reported at different cutoff grades within the base-case conceptual mineral resource-limiting pit shells (Table 14.40) and are presented graphically on grade-tonnage curves (Figure 14.17, Figure 14.18, Figure 14.19, Figure 14.20, and Figure 14.21). It should be noted that this information does not constitute a Mineral Resource Statement and is presented only to demonstrate sensitivity of the deposits to cutoff grade selection.

Table 14.40: Combined Sensitivity to Cutoff Grade

Sulfide Cutoff Grade (g/t Au)	Oxide Cutoff Grade (g/t Au)	Yellow Pine (sulfide)		Hangar Flats (sulfide)		West End (oxide + sulfide)		Historic Tailings (sulfide)		Total (oxide + sulfide)	
		Gold Grade (g/t)	Contained Gold (koz)	Gold Grade (g/t)	Contained Gold (koz)	Gold Grade (g/t)	Contained Gold (koz)	Gold Grade (g/t)	Contained Gold (koz)	Gold Grade (g/t)	Contained Gold (koz)
Indicated											
0.60	0.30	1.80	2,875	1.44	1,199	1.10	1,717	1.16	102	1.44	5,892
0.65	0.35	1.84	2,838	1.50	1,166	1.16	1,645	1.17	102	1.51	5,750
0.70	0.40	1.89	2,799	1.55	1,133	1.23	1,573	1.17	101	1.57	5,606
0.75	0.45	1.93	2,762	1.60	1,103	1.30	1,501	1.19	99	1.63	5,464
0.80	0.50	1.97	2,724	1.66	1,071	1.36	1,438	1.21	96	1.68	5,329
0.85	0.55	2.01	2,684	1.71	1,040	1.42	1,375	1.24	92	1.74	5,191
0.90	0.60	2.05	2,643	1.76	1,011	1.48	1,313	1.26	89	1.79	5,056
Inferred											
0.60	0.30	1.16	438	1.35	404	1.00	368	1.21	6	1.16	1,215
0.65	0.35	1.21	421	1.40	391	1.06	351	1.21	6	1.21	1,169
0.70	0.40	1.26	401	1.46	376	1.10	335	1.22	6	1.26	1,117
0.75	0.45	1.31	380	1.52	363	1.15	317	1.23	6	1.32	1,066
0.80	0.50	1.36	360	1.57	350	1.20	300	1.27	5	1.37	1,016
0.85	0.55	1.41	343	1.63	336	1.23	288	1.30	5	1.42	972
0.90	0.60	1.45	326	1.68	325	1.28	271	1.31	5	1.46	927

Notes:
(1) *Mineral resources are reported in relation to the base-case conceptual pit shell in order to demonstrate potential for economic viability, as required under NI43-101; mineralization lying outside of these pit shells is not reported as a mineral resource. Mineral resources are not mineral reserves and do not have demonstrated economic viability. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated. All figures are rounded to reflect the relative accuracy of the estimate.*

Figure 14.17: Hangar Flats Sulfide Grade versus Tonnage Curves



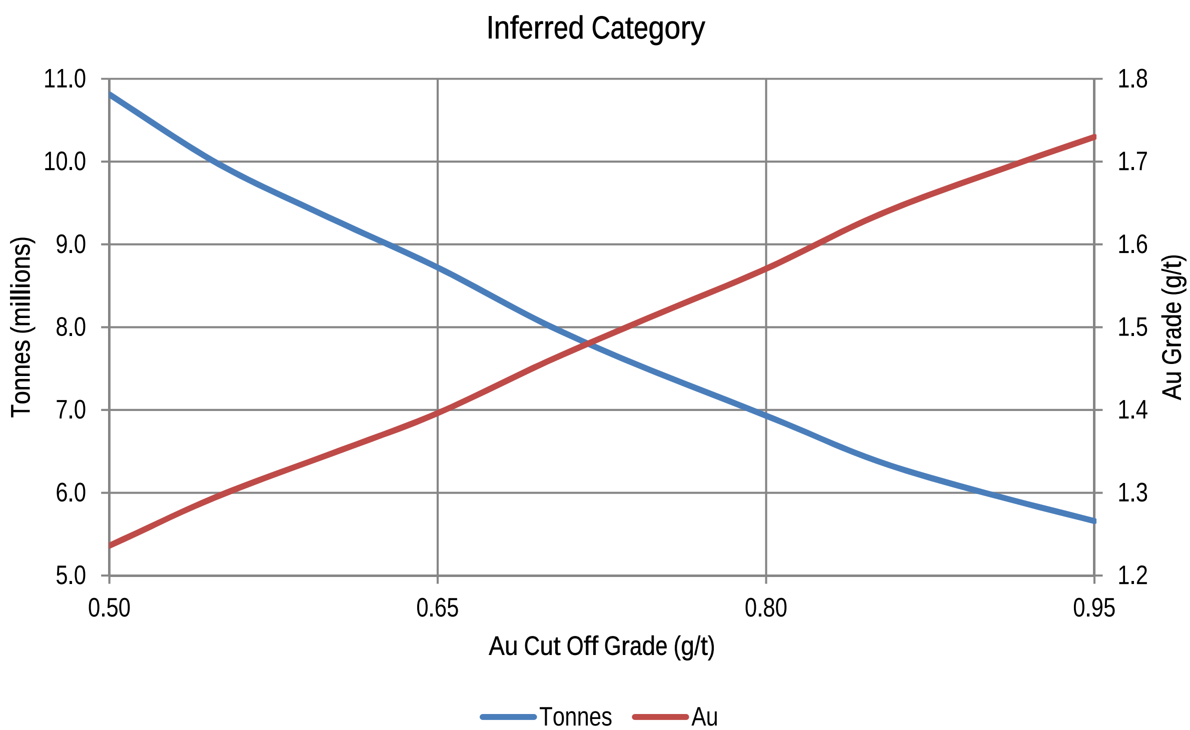
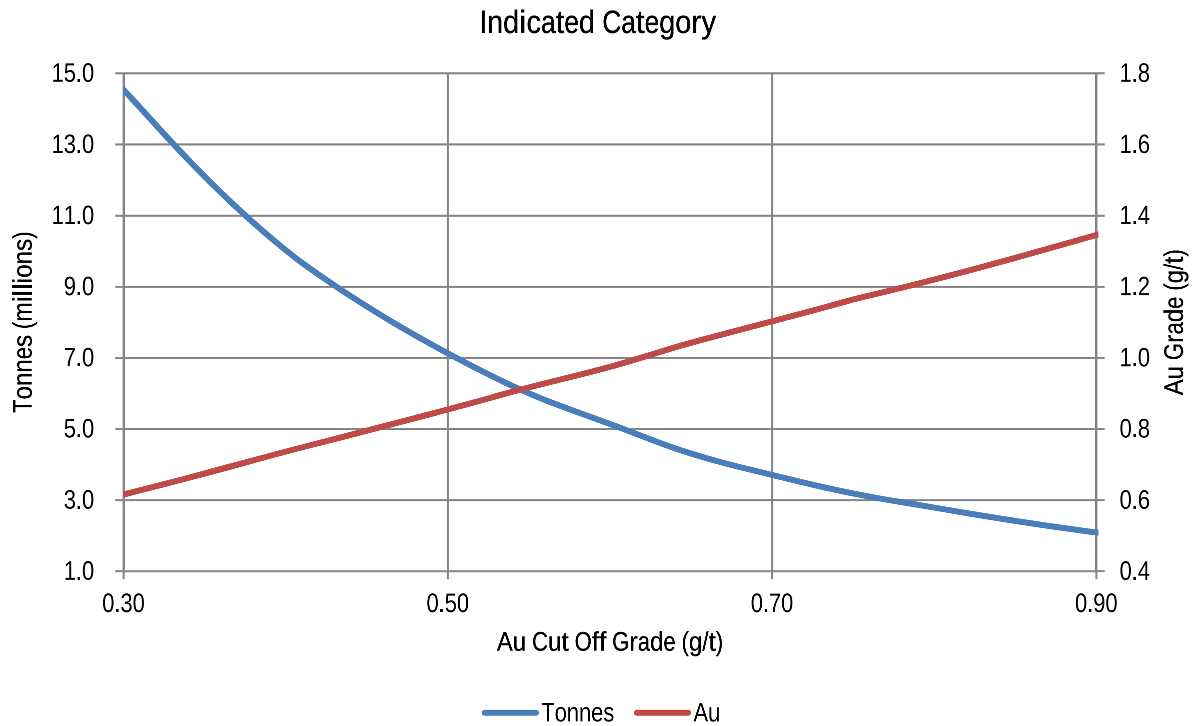


Figure 14.18: West End Oxide Grade versus Tonnage Curves



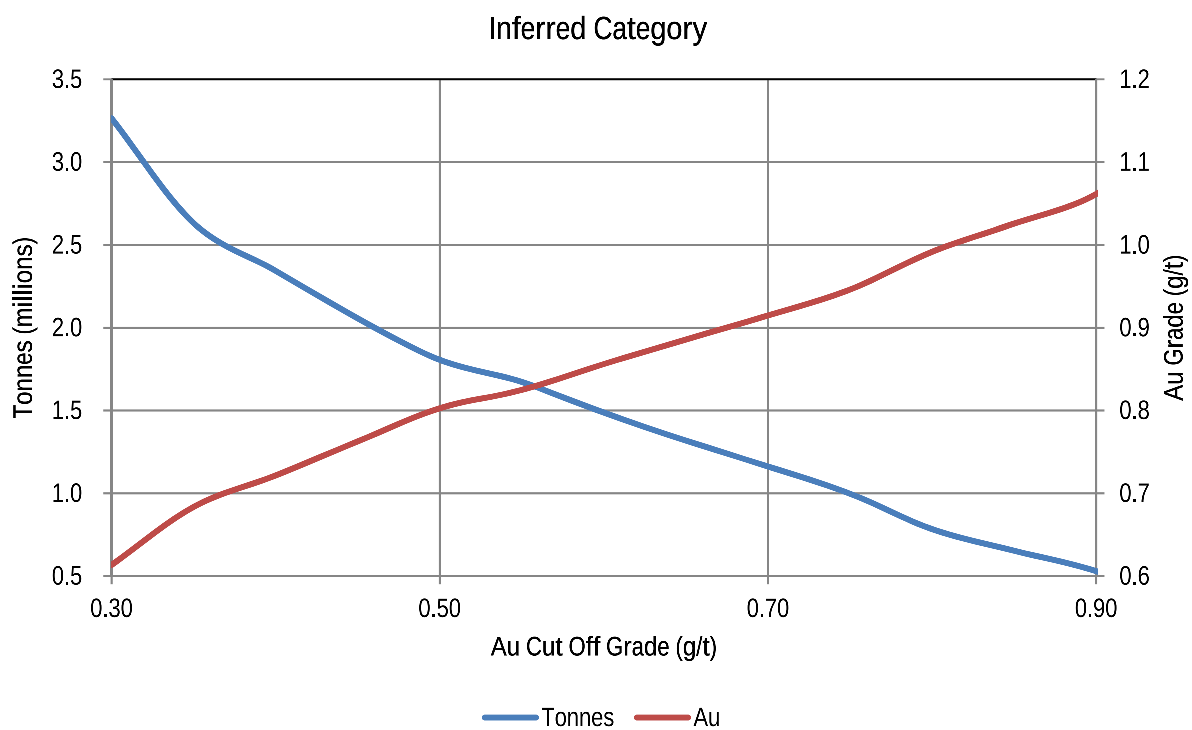
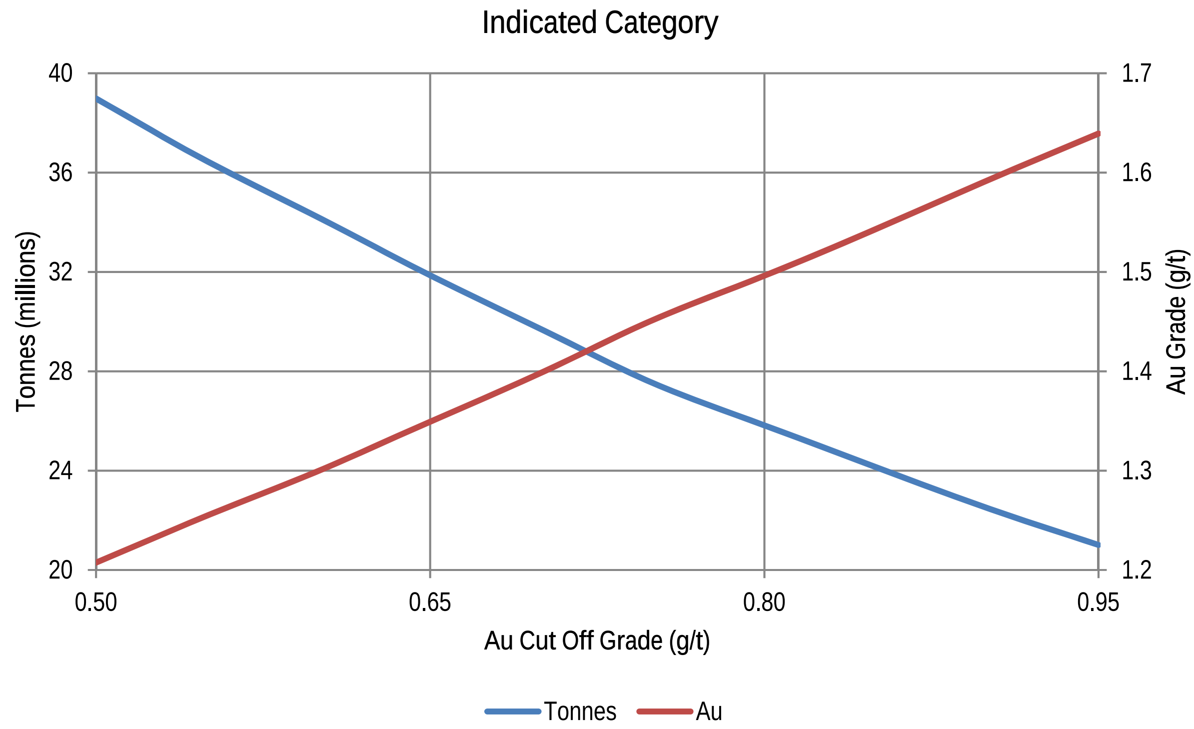


Figure 14.19: West End Sulfide Grade versus Tonnage Curves



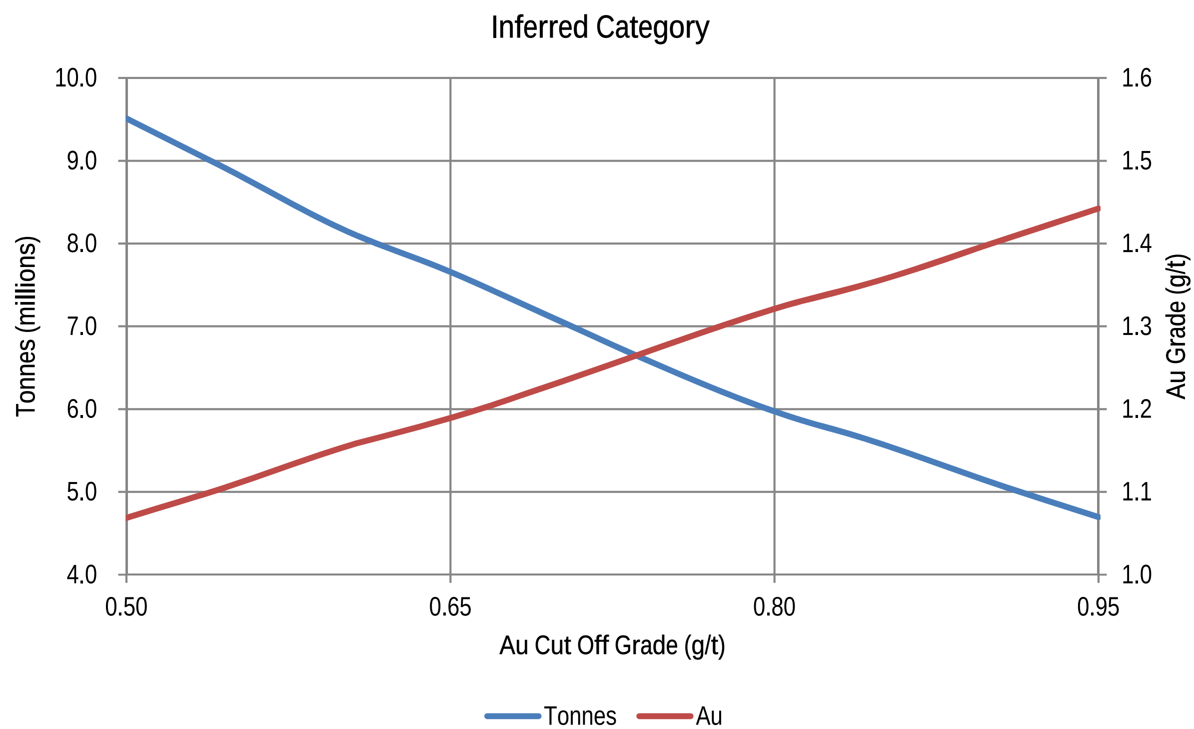
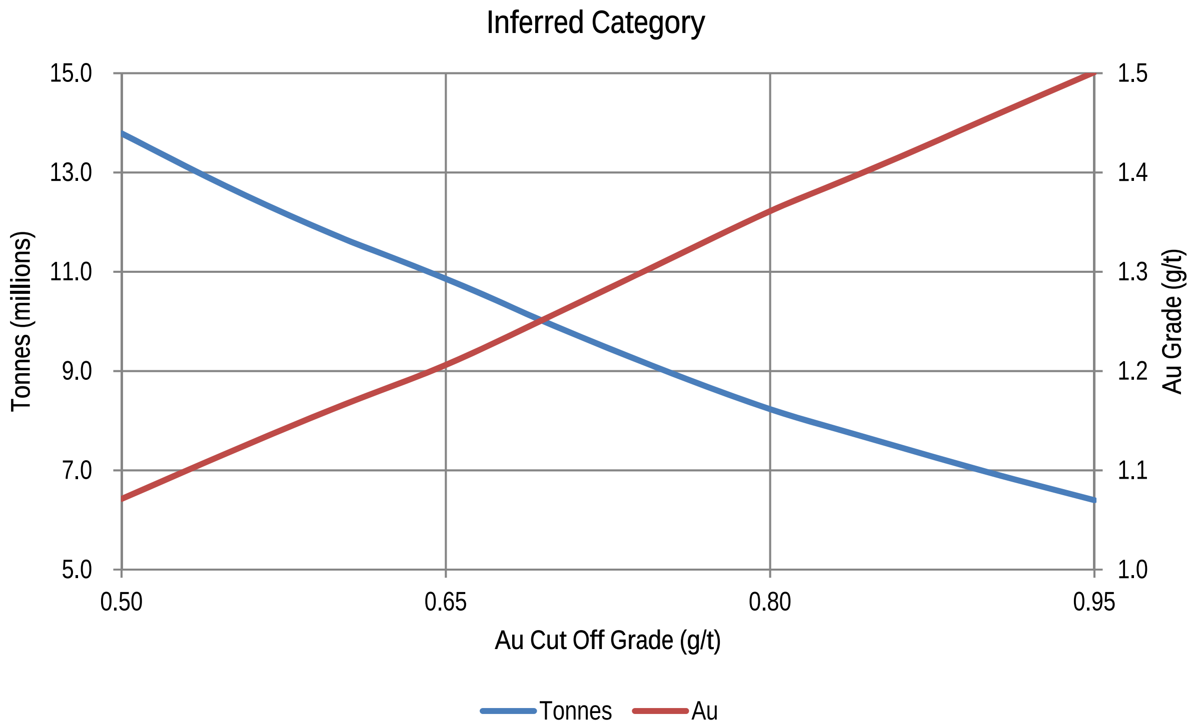


Figure 14.20: Yellow Pine Sulfide Grade versus Tonnage Curves



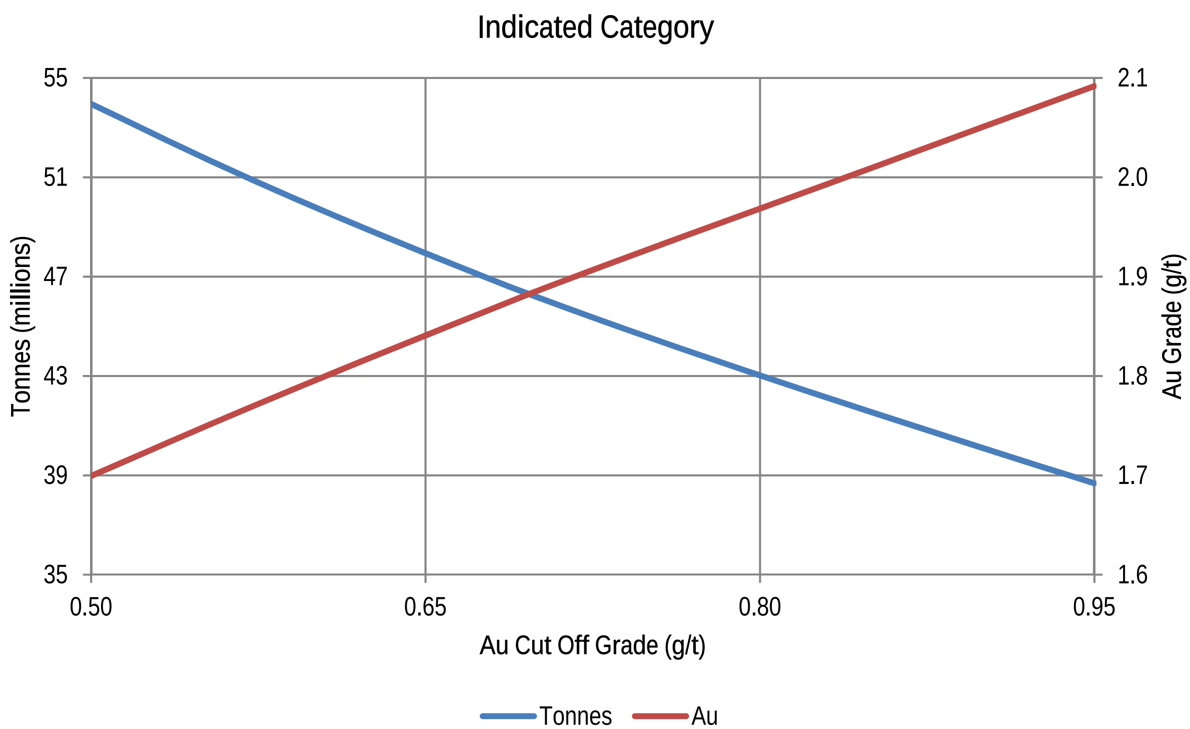
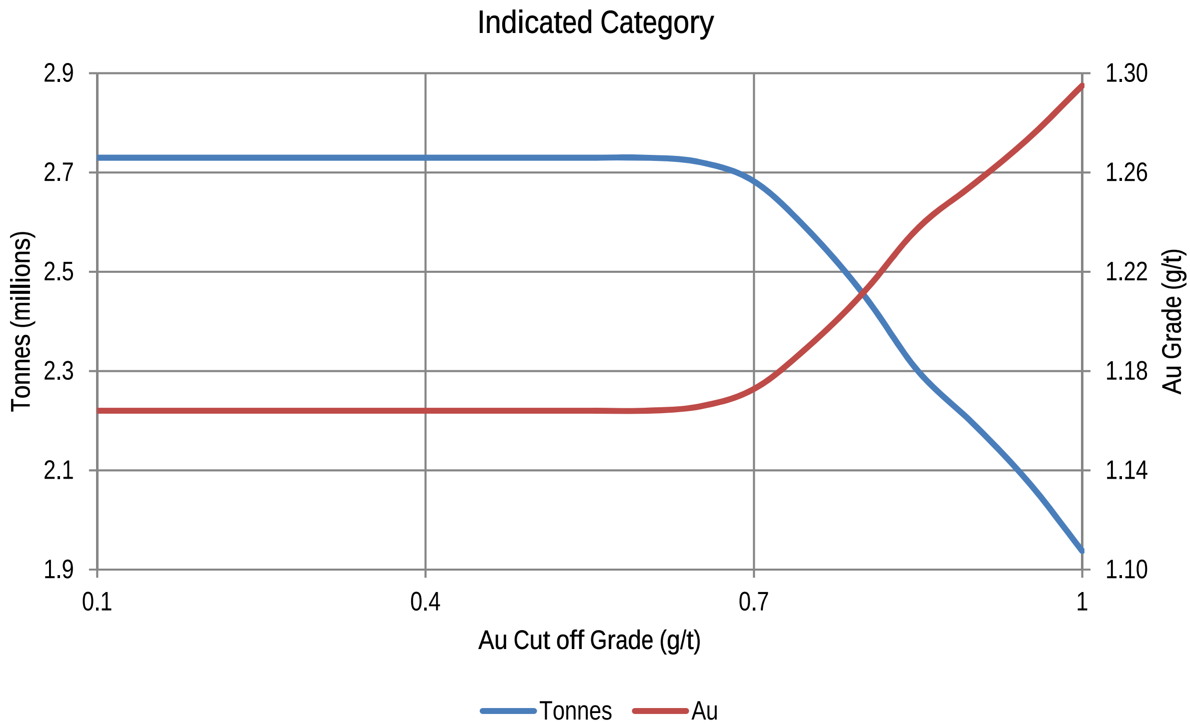
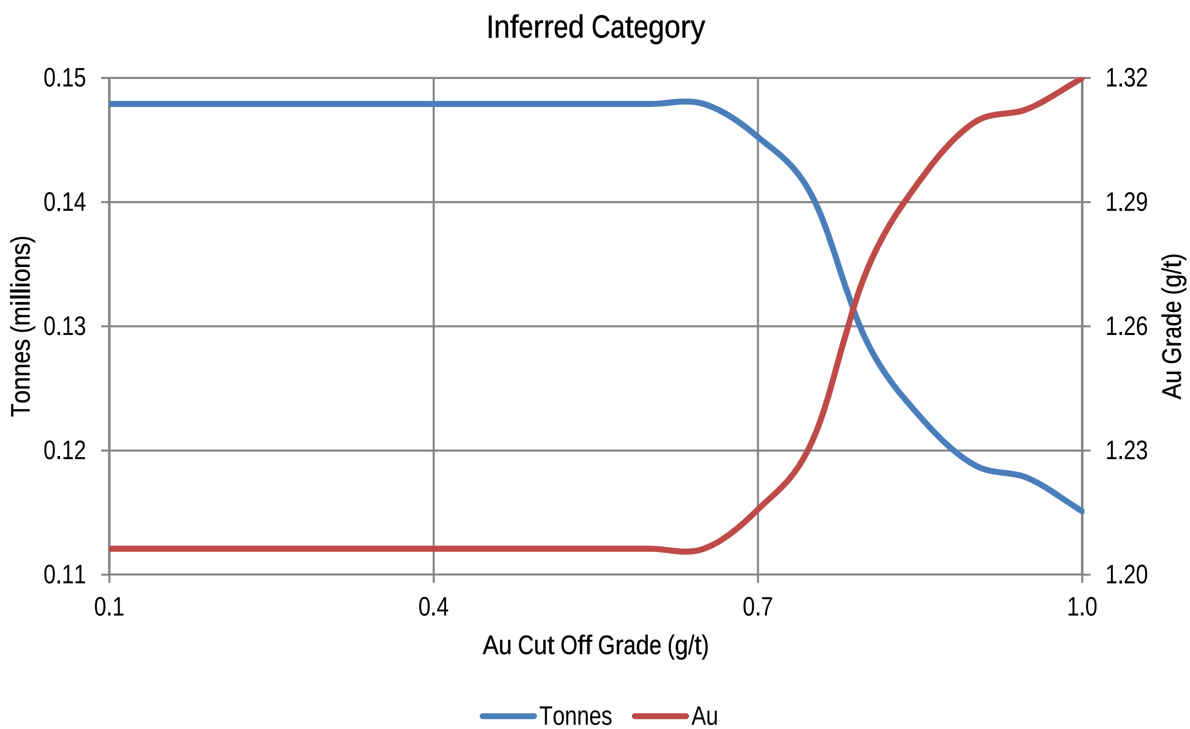


Figure 14.21: Historic Tailings Sulfide Grade versus Tonnage Curves





14.9 COMPARISON OF THE 2012 MINERAL RESOURCE ESTIMATE TO THE CURRENT ESTIMATE

The mineral resource estimates discussed herein incorporate the results of more than 45,000 m of new drilling completed since the cutoff date for the 2012 PEA. The drilling was focused within the Yellow Pine deposit and within a smaller conceptual pit shell at Hangar Flats. Relative to the 2012 PEA, indicated mineral resources for gold increased by 52% in Yellow Pine, 18% at Hangar Flats and remained approximately the same at the West End deposit, where only minimal drilling was completed, as shown in Table 14.41. In addition, the indicated mineral resources for antimony increased by 32% at Yellow Pine and by 22% at Hangar Flats. The West End mineral resource was expanded by drilling, but this increase was largely offset by mineral resource reductions associated with more conservative modeling parameters.

Table 14.41: Percentage Change of the 2012 Mineral Resource Estimate to the Current Estimate

Category	Yellow Pine (sulfide)	Hangar Flats (sulfide)	West End (oxide + sulfide)	Historic Tailings (sulfide)	Total (oxide + sulfide)
Indicated					
Tonnes	65%	24%	6%	100%(5)	34%
Gold Grade	-8%	-5%	-4%	100%(5)	-3%
Silver Grade	325%(6)	228%(6)	100%(4)	100%(5)	407%(6)
Antimony Grade	-18%	-8%	100%(4)	100%(5)	5%
Contained Gold	52%	18%	1%	100%(5)	29%
Contained Silver	604%(6)	308%(6)	100%(4)	100%(5)	579%(6)
Contained Antimony	32%	23%	100%(4)	100%(5)	43%
Inferred					
Tonnes	-72%	-11%	-44%	100%(5)	-55%
Gold Grade	-27%	5%	-7%	100%(5)	-18%
Silver Grade	-1%	5407%(6)	100%(4)	100%(5)	141%
Antimony Grade	-77%	491%	100%(4)	100%(5)	-40%
Contained Gold	-80%	-8%	-48%	100%(5)	-63%
Contained Silver	-72%	4923%(6)	100%(4)	100%(5)	8%
Contained Antimony	-94%	503%	100%(4)	100%(5)	-72%

Notes:
(1) *Mineral resources are reported in relation to conceptual pit shells in order to demonstrate potential for economic viability, as required under NI43-101; mineralization lying outside of these pit shells is not reported as a mineral resource.* ***Mineral resources are not mineral reserves and do not have demonstrated economic viability. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated.*** *All figures are rounded to reflect the relative accuracy of the estimate.*
(2) *2014 Open pit sulfide mineral resources are reported at a cutoff grade of 0.75 g/t Au and open pit oxide mineral resources are reported at a cutoff grade of 0.45 g/t Au. The mineral resources were initially estimated based on the open pit optimization parameters listed in Table 14.33 and a gold selling price of US$1,400/oz, which resulted in an open pit sulfide cutoff grade of approximately 0.55 g/t Au and an open pit oxide cutoff grade approximately 0.35 g/t Au. The 0.75 g/t Au sulfide cutoff grade and 0.45 g/t Au oxide cutoff grade are equivalent to utilizing the cost assumptions stated in Table 14.33 and a gold selling price of approximately $1,000/oz for sulfides and $1,100/oz for oxides.*
(3) *2012 Open pit sulfide mineral resources are reported at a cutoff grade of 0.75 g/t Au and open pit oxide mineral resources are reported at a cutoff grade of 0.42 g/t Au. 2012 mineral resources are reported in relation to the 2012 conceptual pit shells. The cutoff grades and pit shells are based on the open pit optimization parameters discussed in the 2012 PEA (SRK, 2012).*
(4) *Silver and antimony mineral resources at West End were not estimated in 2012 but have been assigned an arbitrary 100% increase.*
(5) *The Historic Tailings mineral resources were not previously estimated but have been assigned an arbitrary 100% increase.*
(6) *The 2012 Yellow Pine and Hangar Flats mineral resource estimates were developed using silver grade shells whereas the 2014 mineral resource estimates were not developed with silver grade shells; the 2014 mineral resource estimation methodology resulted in significant increases in both silver grade and contained silver for the deposits.*

Reductions in the mineral resources occurred primarily where the 2012 PEA estimates allowed extrapolation of grade at depth and around the periphery of the deposits based on the sparse drill data available at the time. Better structural constraints and additional drill hole information incorporated into the updated models now provide for a more conservative estimate, with better controls on gold mineralization. Figure 14.22 through Figure 14.30 depict regions where the new drilling has increased indicated mineral resources within each deposit. The substantial reduction in inferred antimony resources at Yellow Pine relative to the PEA resulted primarily from exclusion of BMC drill holes in the antimony estimate and from use of smaller antimony shells which were constructed using only the higher confidence USBM and post-1973 drill hole data. Numerous BMC underground drill holes located outside of the 2014 antimony shells purportedly intercepted significant antimony mineralization and formed the basis for the majority of inferred antimony resources in 2012. Confirmation drilling completed in 2013 failed to confirm significant antimony mineralization in these areas, suggesting that some antimony mineralization drilled historically from underground may occur in relatively narrow but substantially higher grade zones that would require close-spaced, detailed drilling in order to evaluate the potential for additional antimony mineral resources in these areas.

Figure 14.22: Plan Map of Hangar Flats Showing 2012 and 2014 Mineral Resource Classification

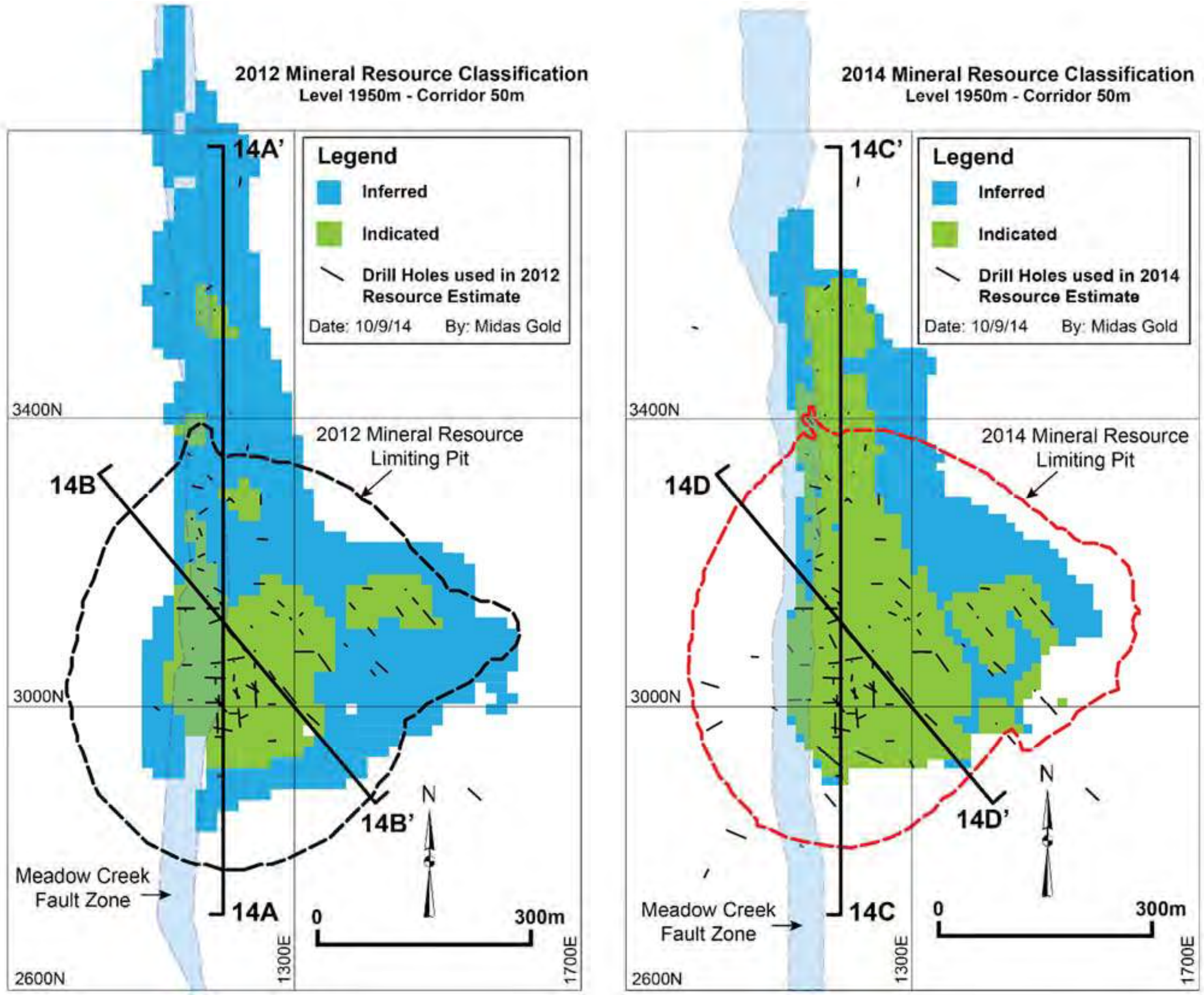
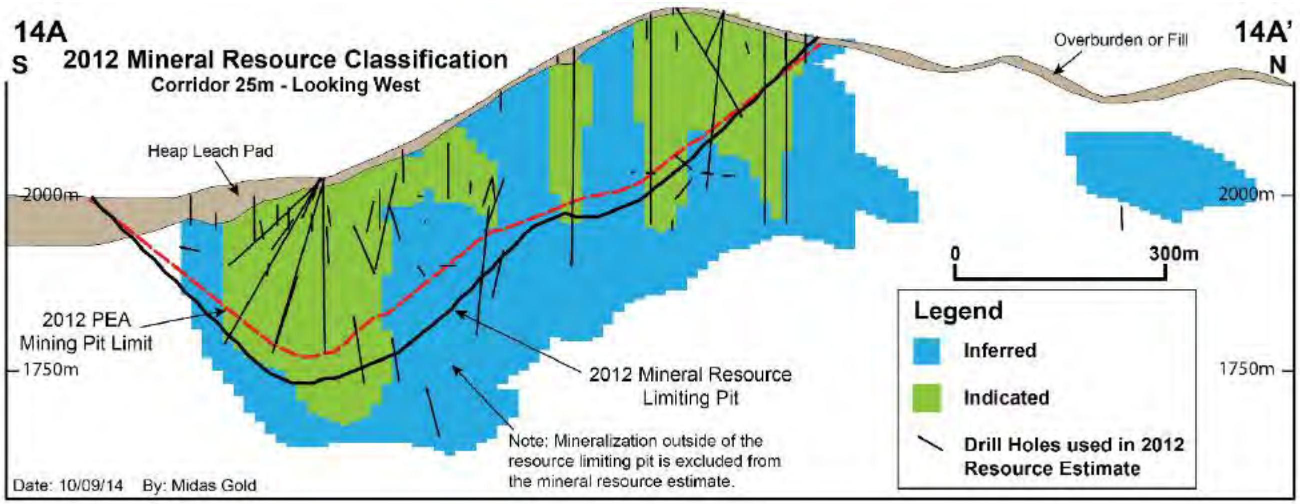


Figure 14.23: Long Section of Hangar Flats Showing 2012 and 2014 Mineral Resource Classification



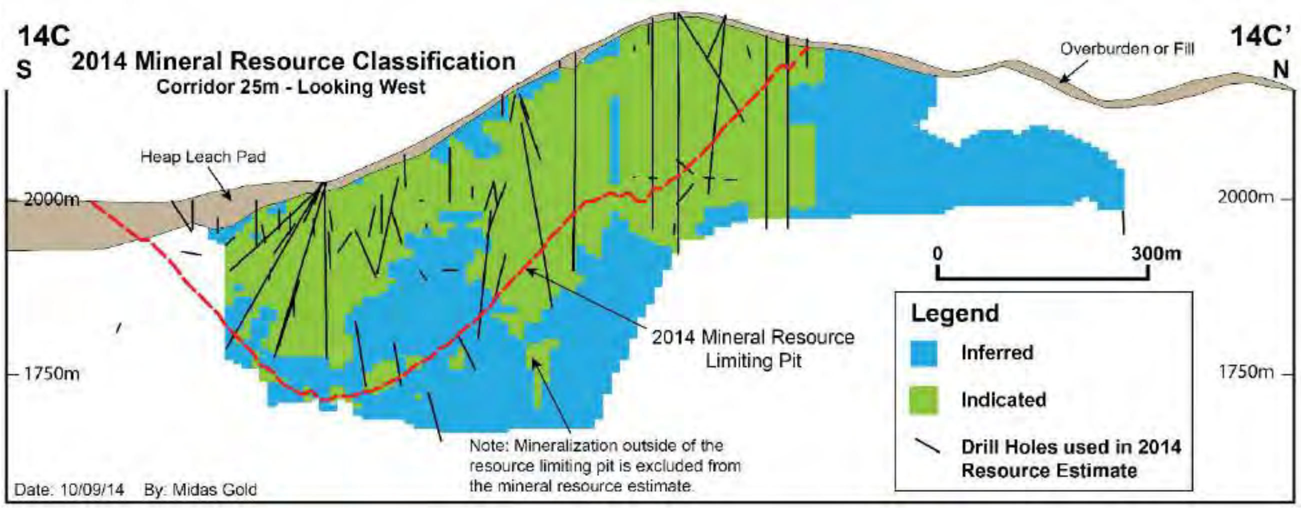
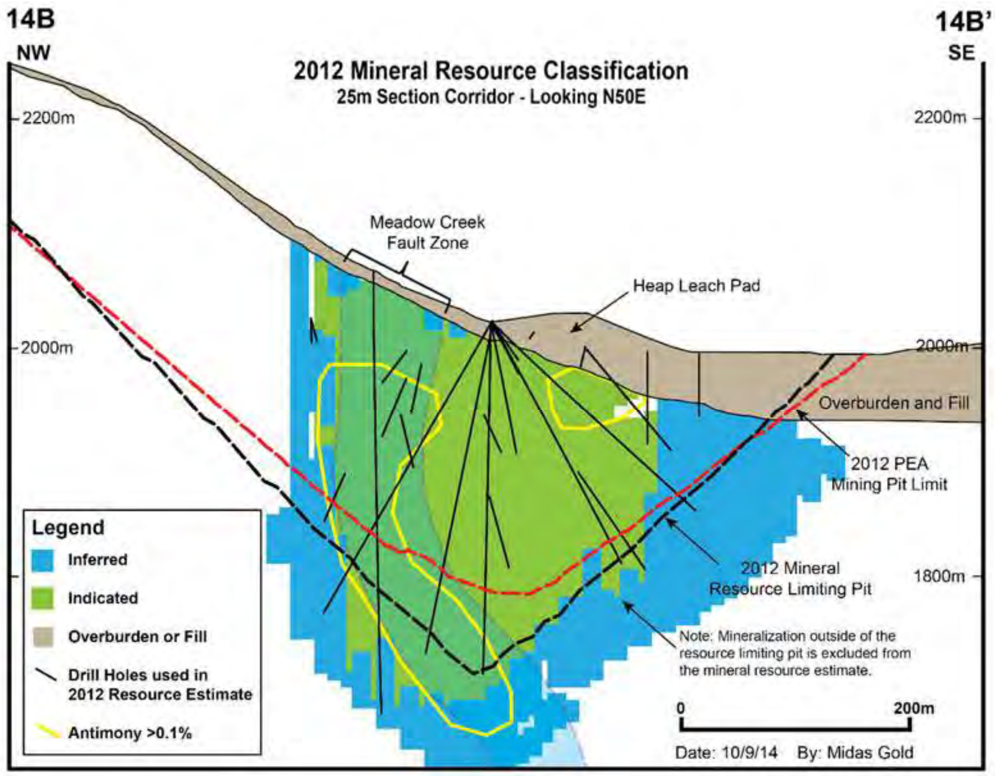


Figure 14.24: Cross Section of Hangar Flats Showing 2012 and 2014 Mineral Resource Classification



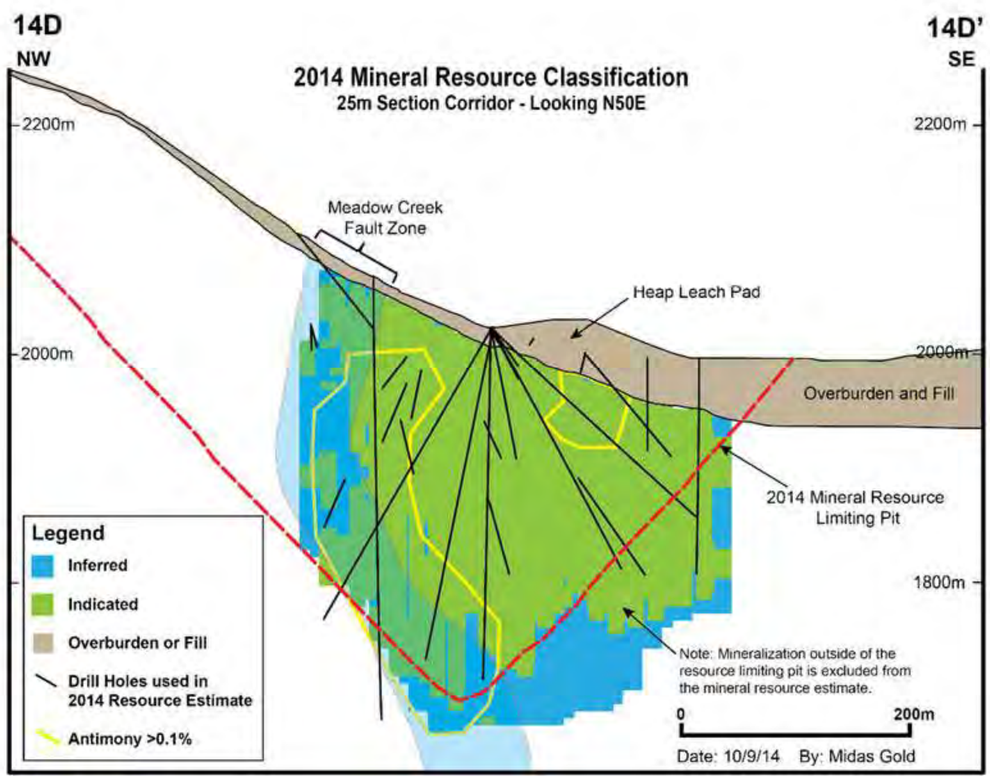


Figure 14.25: Inclined Plan Map of West End Showing 2012 and 2014 Mineral Resource Classification

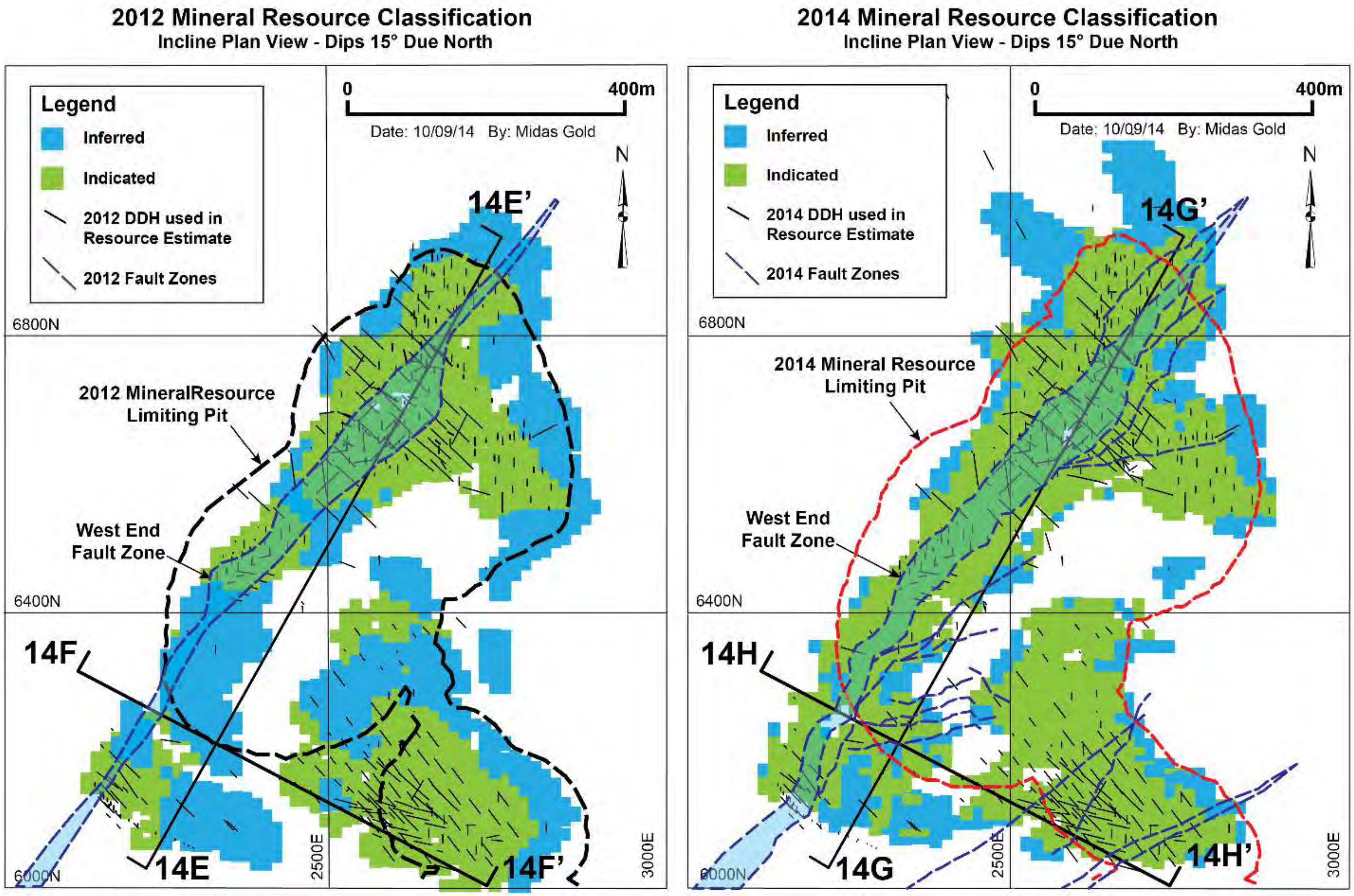


Figure 14.26: Long Section of West End Showing 2012 and 2014 Mineral Resource Classification

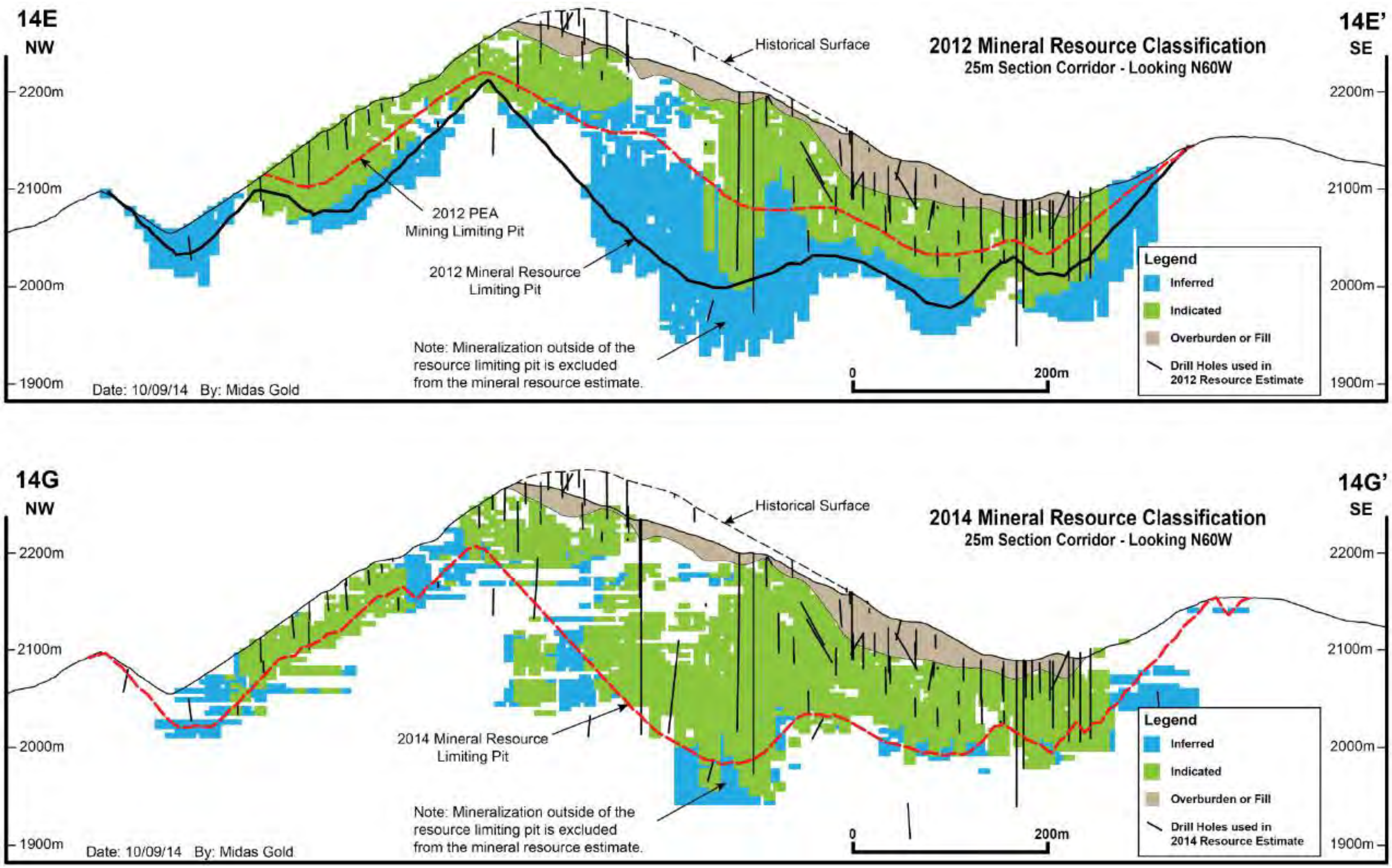


Figure 14.27: Cross Section of West End Showing 2012 and 2014 Mineral Resource Classification

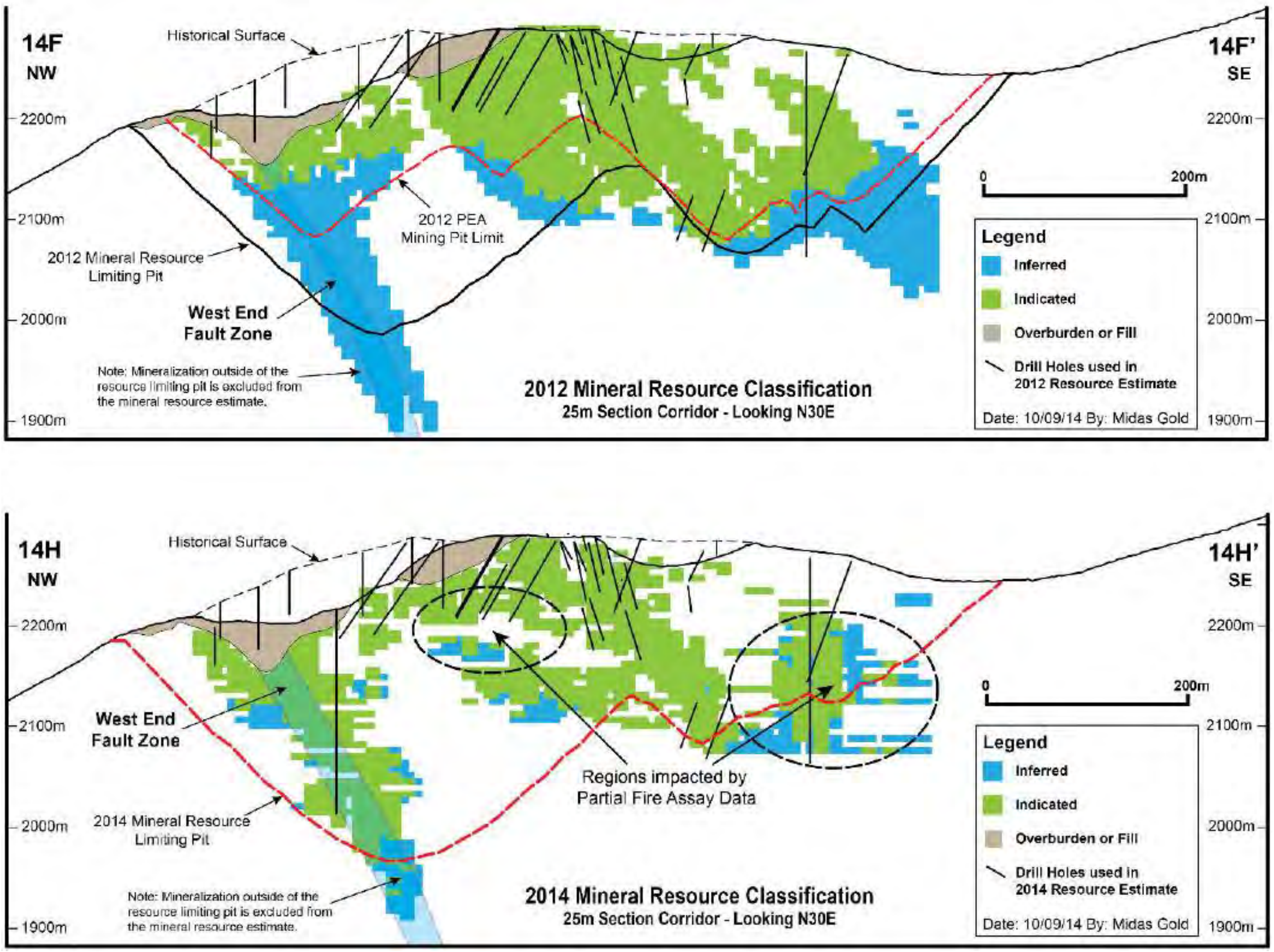


Figure 14.28: Plan Map of Yellow Pine Showing 2012 and 2014 Mineral Resource Classification

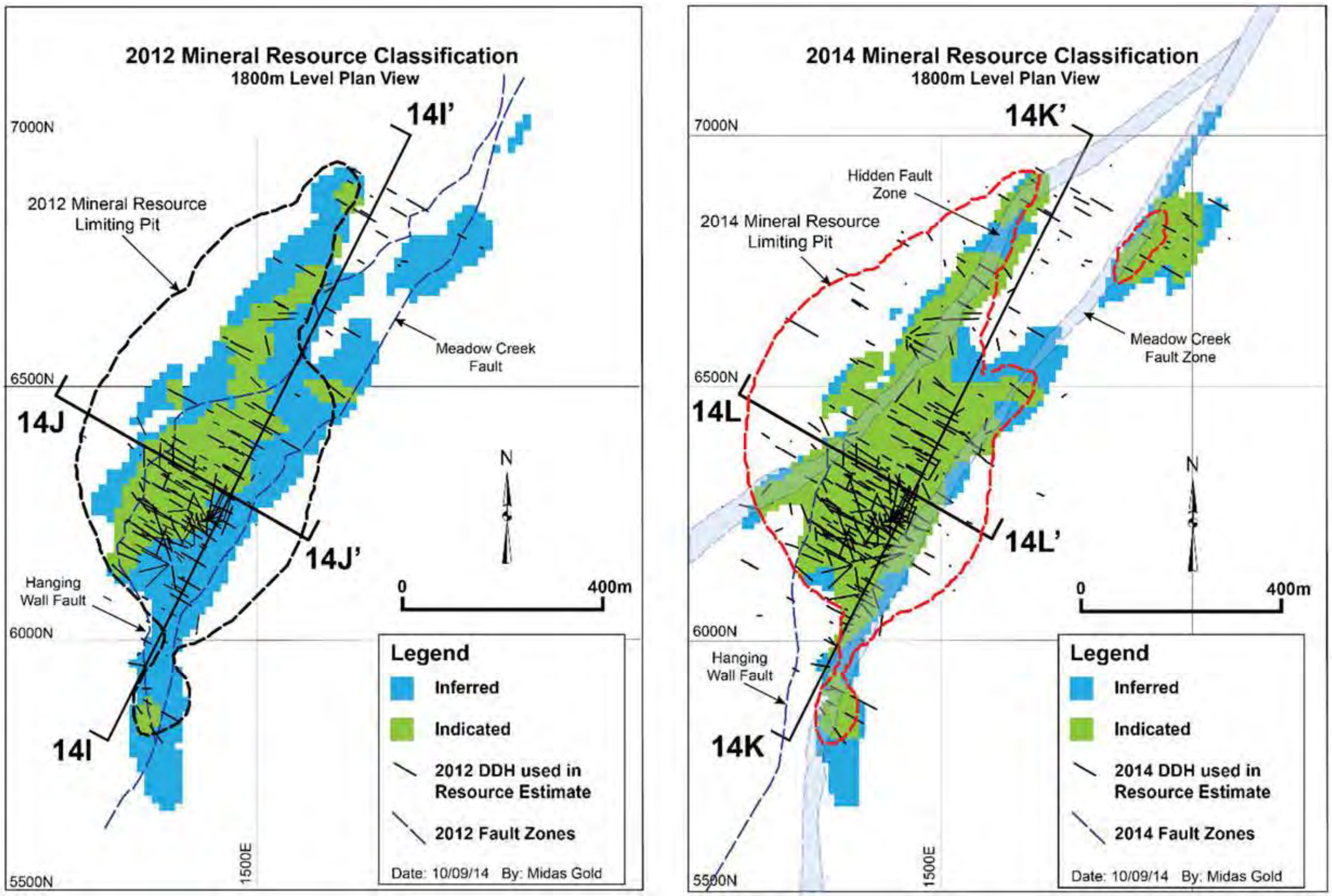


Figure 14.29: Long Section of Yellow Pine Showing 2012 and 2014 Mineral Resource Classification

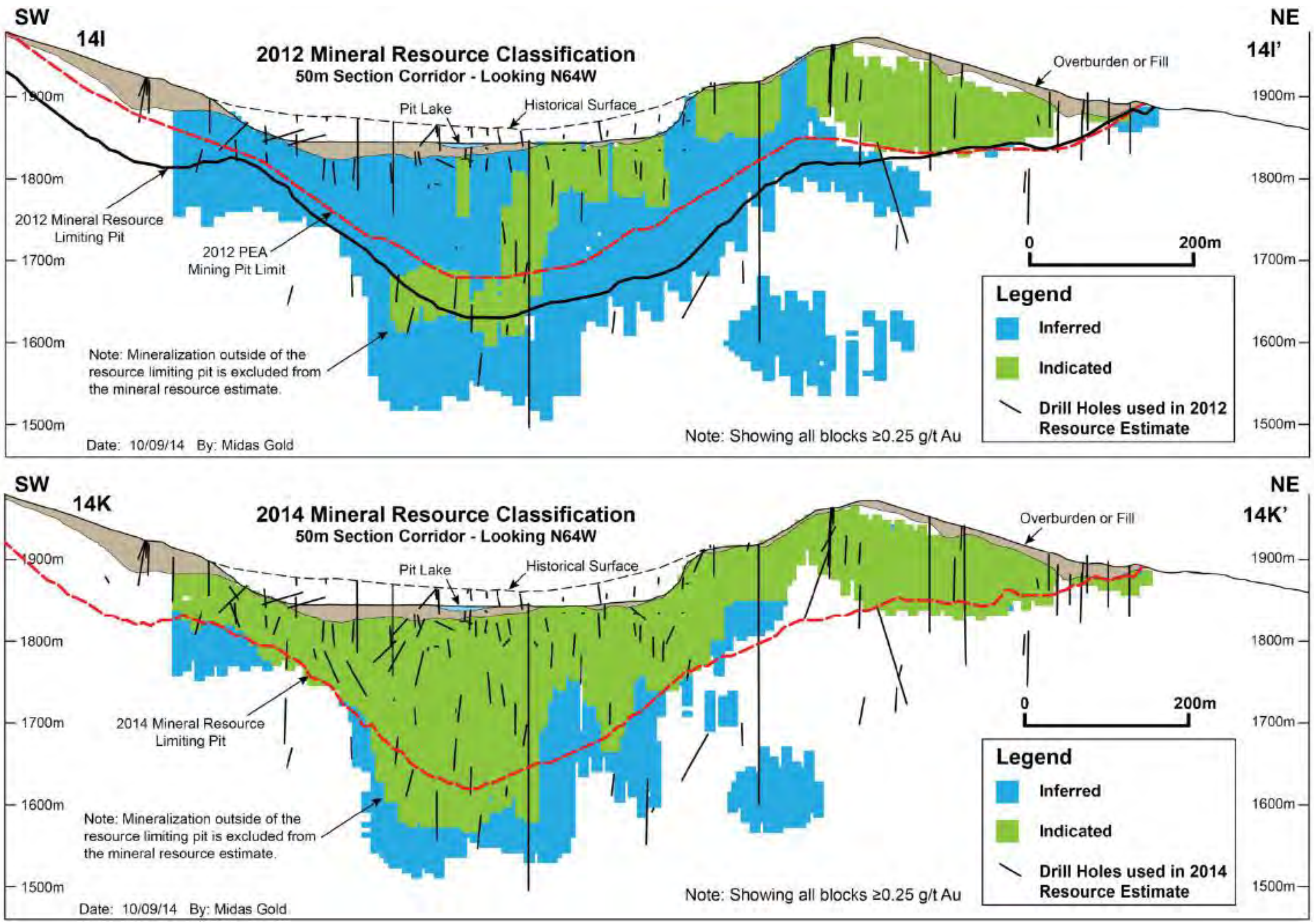
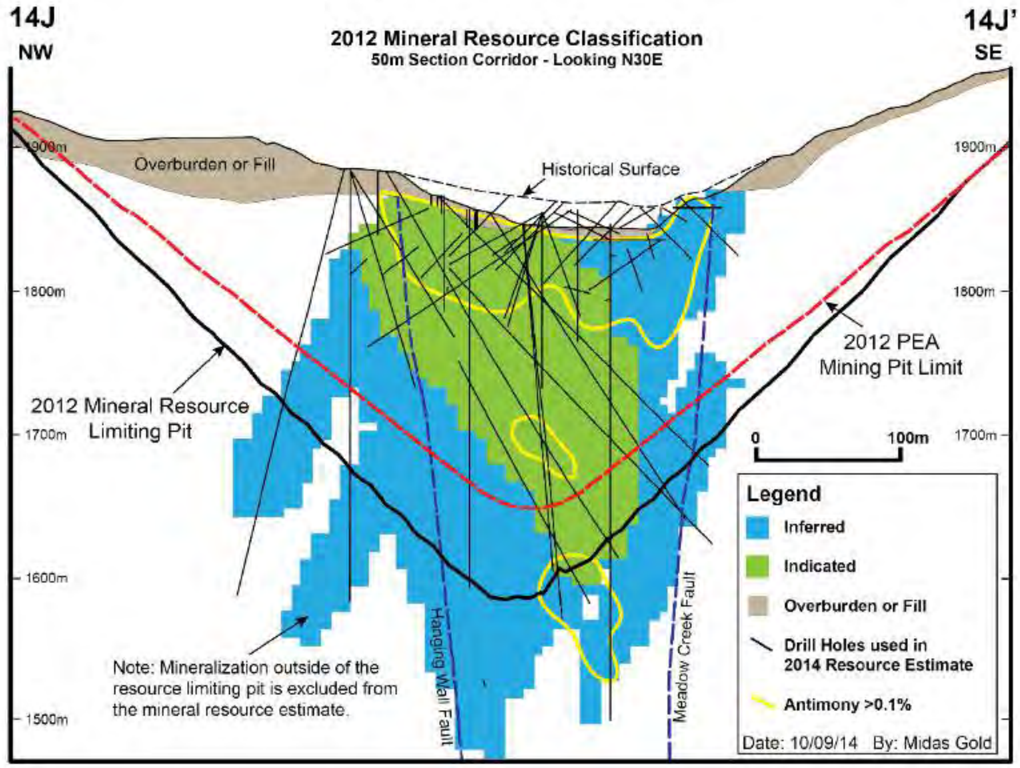
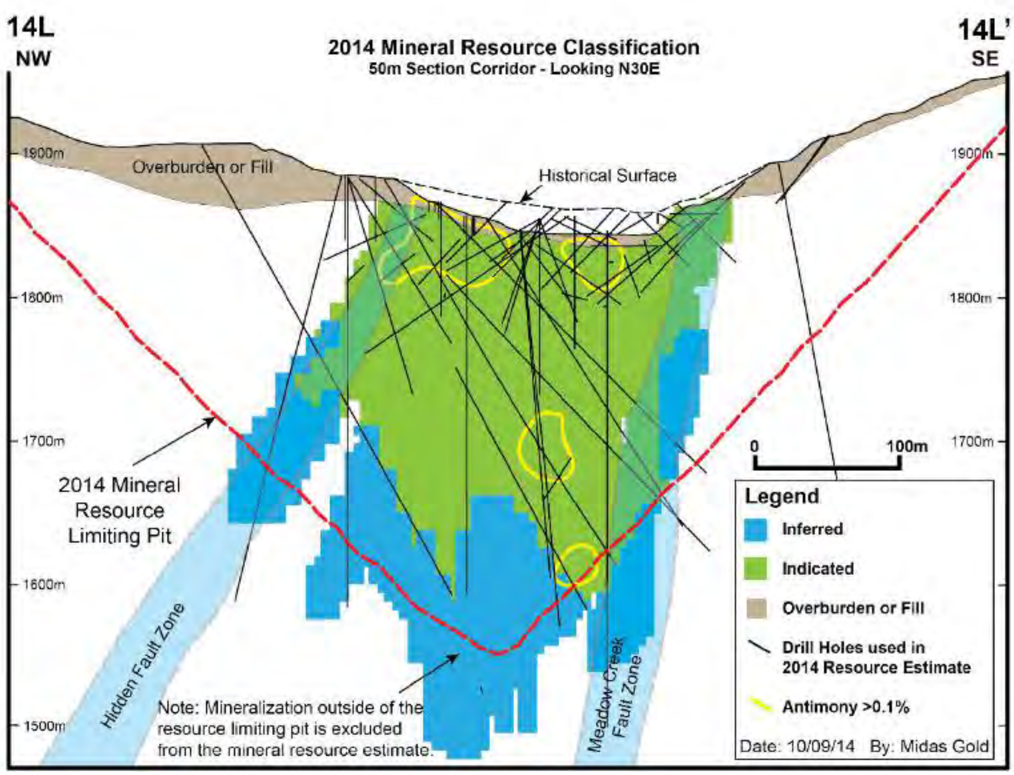


Figure 14.30: Cross Section of Yellow Pine Showing 2012 and 2014 Mineral Resource Classification





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15 MINERAL RESERVE ESTIMATES

15.1 INTRODUCTION

This section describes the Mineral Reserve estimation methodology, summarizes the key assumptions used, and presents the Mineral Reserve estimates for the Project. The qualified person (QP) for the estimation of the Mineral Reserve was John M. Marek, P.E. of Independent Mining Consultants, Inc. The Mineral Reserve estimates reported herein are a reasonable representation of the Mineral Reserves within the Project at the current level of analysis. The Mineral Reserves were estimated in conformity with generally accepted Canadian Institute of Mining and Metallurgy (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" and are reported in accordance with the Canadian Securities Administrators' NI 43-101. Mr. Marek has reviewed the risks, opportunities, conclusions and recommendations summarized in Sections 25 and 26, and he is not aware of any unique conditions that would put the Stibnite Gold Mineral Reserve at a higher level of risk than any other North American developing projects.

The Mineral Reserve is the total of all Probable category material (there is no material in the Proven category) that is planned for production. The mine plan that is presented in Section 16 details the production of that Mineral Reserve. No low-grade stockpiles were considered in the mine plan for a lack of an acceptable location; therefore, the Mineral Reserve is established by tabulating the Indicated Mineral Resources that are planned for processing in each year which equates to the Probable Mineral Reserve. The final pit design and internal phase designs that contain the Mineral Reserve were guided by the results of the floating cone algorithm.

15.2 FLOATING CONES

The floating cone algorithm is a tool for phase design guidance. The algorithm applies approximate costs and recoveries along with approximate open pit slope angles to establish theoretical economic breakeven pit wall orientations.

Economic input applied to the cone algorithm is necessarily preliminary as it is one of the first steps in the development of the mine plan. The cone geometries should be considered as approximate as they do not assure access or working room. The important result of the cones is the relative change in geometry between cones of increasing metal prices. Lower metal prices result in smaller pits containing materials with higher margins, which provide guidance to the design of the initial phase designs. The change in pit geometry as metal prices are increased indicates the best directions for the succeeding phase expansions to the ultimate open pit.

Cones were floated for the Yellow Pine (YP), Hangar Flats (HF) and West End (WE) deposits using gold prices ranging from $200 to $1,500 per ounce. The costs and recoveries used for input were based on the PEA results and are provided in Table 15.1 through Table 15.4, inclusively. Net of Process Revenue (NPR), defined as Net Smelter Return (NSR) less process plant operating expenditures (OPEX) and general and administrative costs (G&A), was calculated on a block-by-block basis in dollars per ton of ore ($/st ore) to indicate the value of a block.

NPR = NSR – Process Plant OPEX – Site G&A

Designing with NPR was chosen because Stibnite Gold Project Mineral Resources are poly-metallic with separate process streams that have distinct differences in processing costs. It was necessary to account for processing costs in order to determine the appropriate ore type category of a block. Mining costs are not included in the calculation of NPR because the mining cost will be essentially the same for an ore block regardless of the process designation.

Table 15.1 Process Plant Recoveries for Floating Cones

Process	Metal	YP	HF	WE
Oxide Recovery	Gold	80%	80%	CN:FA x 100%
	Silver	80%	80%	80%
Sulfide Flotation Recovery	Gold High Sb	88%	89%	N/A
	Gold Low Sb	93%	92%	$(1.7366 \times CN{:}FA^3 - 2.5676 \times CN{:}FA^2 - 0.0858 \times CN{:}FA + 0.9231) \times 100\%$
	Silver High Sb	68%	68%	N/A
	Silver Low Sb	85%	80%	N/A
Transition Recovery	Gold	N/A	N/A	92%
	Silver	N/A	N/A	80%
POX Recovery	Gold	98%	98%	98%
	Silver	7%	7%	7%
Recovery to Antimony Concentrate	Gold	1.5%	1.5%	1.5%
	Silver	12%	12%	12%
	Antimony	80%	80%	80%
Antimony Con Grades	Antimony	50%	50%	50%
	Moisture	10%	10%	10%

Note: CN:FA is the ratio of cyanide soluble gold / fire assay gold.

Table 15.2 Payables and Transport Costs for Floating Cones

Off Site Costs and Payables	Item	Unit	Value
Payables for Dore	Gold	%	99.0
	Silver	%	95.0
Dore Ref/Transport Cost	Gold	$/paid oz	8.00
	Silver	$/paid oz	0.50
Smelter Payables for Antimony Concentrate	Gold (if Au concentrate grade > 0.292 oz/st)	%	60
	Silver (if Ag concentrate grade > 2.92 oz/st)	%	30
	Antimony	%	65
TC/RC Costs for Antimony Concentrate		$/wet st	141.52
		$/dry st	155.67

Table 15.3 Process Plant and G&A Costs for Floating Cones

Mineral Resource Type	Process Plant and G&A Costs
Oxide	$11.68/st ore
High Antimony Sulfide	$16.93/st ore
Low Antimony Sulfide	$18.43/st ore
Low Antimony Transition	$21.70/st ore

Table 15.4 Other Open Pit Optimization Parameters

Item	Parameter	Value
Royalties	Dore Produced Onsite and Concentrate Shipped Offsite	1.7% Net of Smelter on Gold
Mining Costs	All Pits	$1.73 / st of material
Bench Discounting	Yellow Pine Hangar Flats West End	0.8% / bench 1.1% / bench 1.1% / bench
Open Pit Slope Angles	Variable by open pit sector	Variable based on recommendations from Strata, A Professional Services Corporation (Strata, 2014); flattened to account for haul roads.

Bench discounting was considered in the development of the floating cones as a way to incorporate the time value of money into the pit optimization algorithm. Using the estimated vertical mining rate, an annual discounting rate can be approximated with bench discounting. For example, Yellow Pine had an estimated 12 benches per year mining rate based on a preliminary schedule. Assuming a 10% discount rate per year, each bench is discounted by 0.8% (10% / 12 benches) starting from the pit crest. Since waste rock is stripped ahead of ore to assure ore release, the time value of delayed ore can be compared against the preceding cost of waste stripping. The bench discounting rates are provided in Table 15.4.

Cones were generated by allowing only Indicated Mineral Resource blocks (there are no Measured Mineral Resources) to contribute positive economic value. Confidence classification was based on gold-only estimation. Although the NPR values calculated for each block included value for silver and antimony, if a block was uneconomical based on gold content alone, it was treated as waste. This prevented blocks from contributing positive economics because of their additional silver and antimony content, ensuring that silver and antimony are treated as true byproducts.

Within the floating cones, blocks were categorized by the process that generated the greatest NPR value. If processing a block produced a negative NPR value, the block was considered waste. The block categorization bins by ore deposit used for floating cones are explained below:

Yellow Pine and Hangar Flats blocks were designated as follows:

- High Antimony Sulfide Ore: An antimony concentrate would be produced followed by a gold bearing sulfide concentrate. The sulfide concentrate would be processed onsite through pressure oxidation and cyanide leaching.

or

- Low Antimony Sulfide Ore: Only a gold bearing sulfide concentrate would be produced. The sulfide concentrate would be processed onsite through pressure oxidation and cyanide leaching.

or

- Waste Rock

West End Blocks were designated as follows:

- Low Antimony Sulfide Ore: A gold bearing sulfide concentrate would be produced. The sulfide concentrate would be processed onsite through pressure oxidation and cyanide leaching.

or

- Oxide Ore: Gold would be recovered through whole ore cyanide leaching

or

- Transition Ore: A Gold bearing sulfide concentrate would be produced. The sulfide concentrate would be processed onsite through pressure oxidation and cyanide leaching. Additional gold would be recovered through cyanide leaching of the tails.

or

- Waste Rock

15.3 GUIDANCE CONE SELECTION AT YELLOW PINE

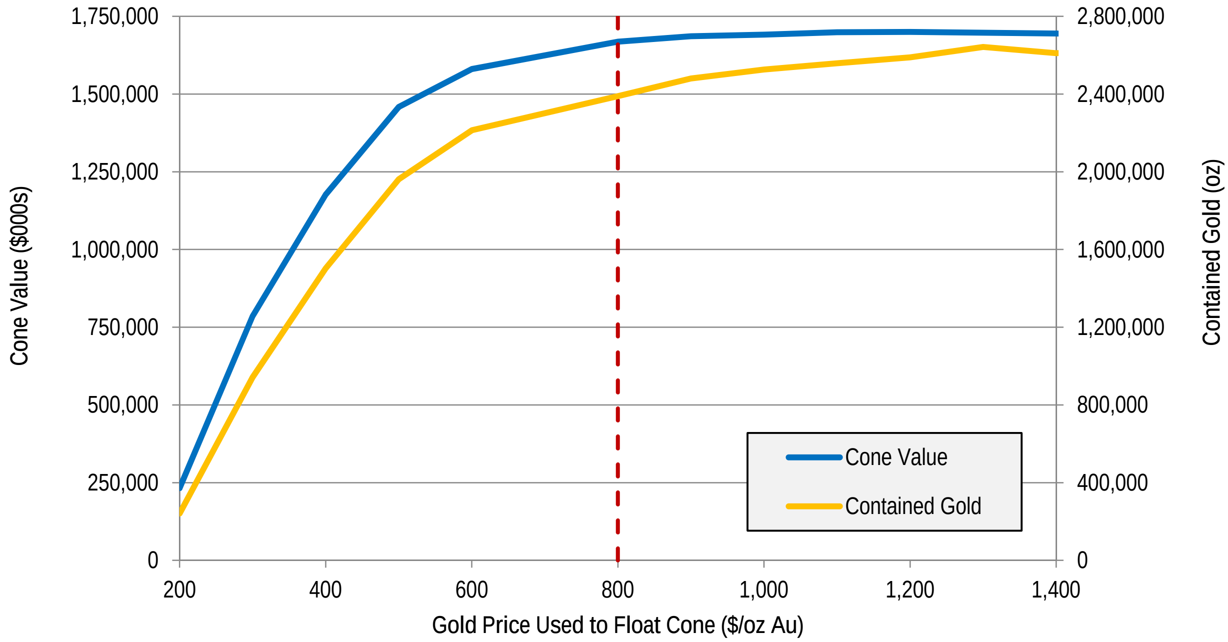
A range of cone geometries were developed for the Yellow Pine deposit by varying the gold price between $200/oz. and $1,500/oz. Costs were held constant in each case and a floating cone pit geometry was established at each assumed metal price. Floating cones establish the pit wall location on a breakeven economic basis. The cones thus derived were then evaluated at a $1,200/oz gold price without changing the size of the cone. The purpose of this work was to see if there was a point of diminishing returns as the cone size increased where little value is added by increasing the pit size. This happens at the $800/oz Au cone; therefore the final pit of Yellow Pine was designed to contain the ore within the $800/oz Au cone. The additional value contained within the pits that were generated at metal prices above $800/oz was incrementally marginal compared to the $800/oz geometry. The benefit of mining a larger pit would become more marginal or even negative once the mine schedule is completed and the value of the pit is evaluated on a discounted basis. The tonnage curves of the cones between $200 and $1,500/oz Au are given in Table 15.5 on and Figure 15.1.

Table 15.5: Yellow Pine Comparison of Increasing Cone Sizes for Constant $1,200/oz Gold Price

Au [1] Price	Mineral Resource (kst)	Net of Process ($/st)	Au (oz/st)	Sb (%)	Ag (oz/st)	Waste Rock (kst)	Total (kst)	Strip Ratio (waste/ore)	Net of Process ($000s)	Mining Cost ($000s)	Value ($000s)	Contained Au (oz)
1,500	49,608	41.66	0.053	0.059	0.043	167,928	217,536	3.39	2,066,669	376,337	1,690,332	2,629,224
1,400	49,247	41.81	0.053	0.059	0.043	161,295	210,542	3.28	2,059,017	364,238	1,694,779	2,610,091
1,300	48,928	41.95	0.054	0.059	0.043	156,242	205,170	3.19	2,052,530	354,944	1,697,586	2,642,112
1,200	47,944	42.34	0.054	0.060	0.043	142,793	190,737	2.98	2,029,949	329,975	1,699,974	2,588,976
1,100	47,383	42.52	0.054	0.061	0.044	135,162	182,545	2.85	2,014,725	315,803	1,698,922	2,558,682
1,000	45,939	43.17	0.055	0.062	0.045	122,965	168,904	2.68	1,983,187	292,204	1,690,983	2,526,645
900	45,090	43.53	0.055	0.063	0.045	114,813	159,903	2.55	1,962,768	276,632	1,686,136	2,479,950
800	43,430	44.17	0.055	0.065	0.046	100,947	144,377	2.32	1,918,303	249,772	1,668,531	2,388,650
600	38,835	45.57	0.057	0.070	0.049	70,637	109,472	1.82	1,769,711	189,387	1,580,324	2,213,595
500	33,817	47.56	0.058	0.077	0.053	52,718	86,535	1.56	1,608,337	149,706	1,458,631	1,961,386
400	24,633	51.39	0.061	0.096	0.064	27,102	51,735	1.10	1,265,890	89,502	1,176,388	1,502,613
300	14,061	59.03	0.067	0.141	0.081	11,861	25,922	0.84	830,021	44,845	785,176	942,087
200	3,256	74.67	0.074	0.331	0.150	3,203	6,459	0.98	243,126	11,174	231,951	240,944

Note:
(1) The gold price in the first column is the gold price that was used to generate the cone geometry. The remaining columns report the results of geometries being re-evaluated using a $1,200/oz gold price.

Figure 15.1 Yellow Pine Comparison of Various Cone Sizes at $1200/oz Au



15.4 GUIDANCE CONE SELECTION AT HANGAR FLATS AND WEST END

Hangar Flats and West End were evaluated in the same manner as Yellow Pine. The curves of the increasing cone sizes evaluated at $1,200/oz Au for Hangar Flats and West End are provided on Figure 15.2 and Figure 15.3, respectively.

Figure 15.2 Hangar Flats Comparison of Various Cone Sizes at $1,200/oz Gold

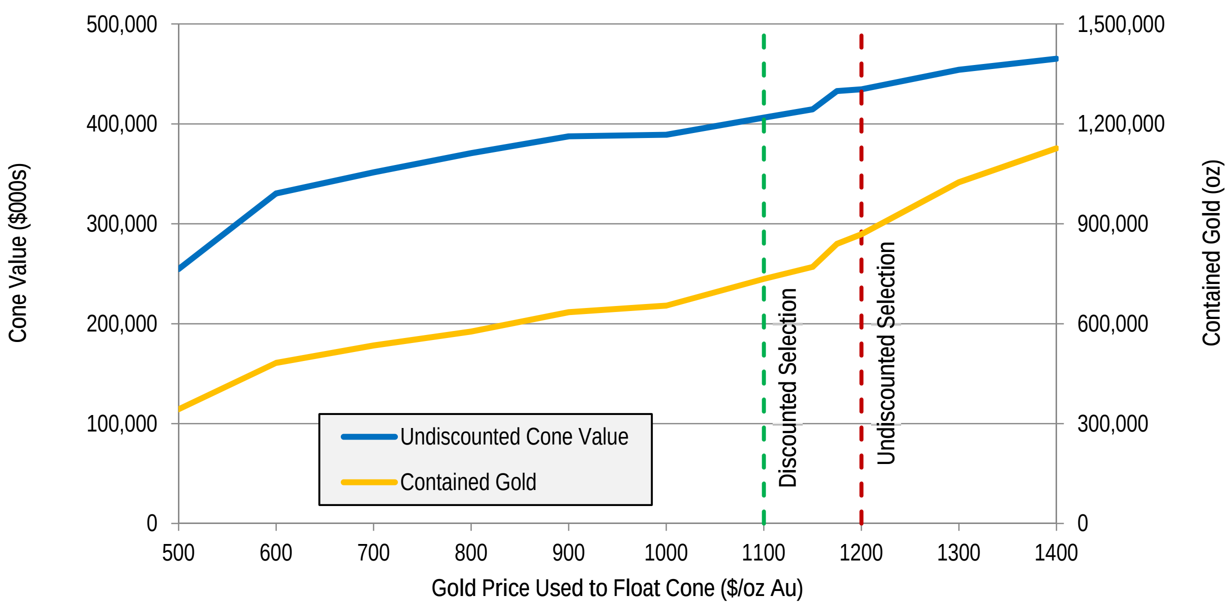
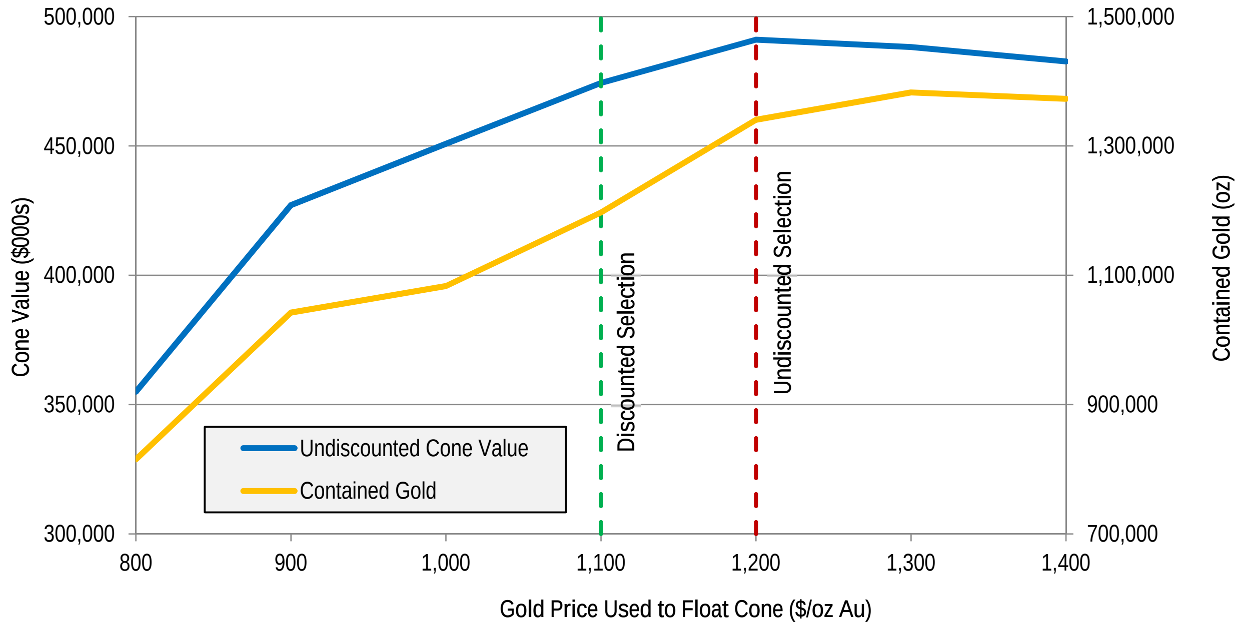


Figure 15.3 West End Comparison of Various Cone Sizes at $1,200/oz Gold



The $1,200/oz Au floating cone appeared to be the appropriate size for both Hangar Flats and West End; however, Hangar Flats and West End come later in mine life than Yellow Pine and additional work was needed to ensure the proper sized floating cone was used for phase design guidance. Significant ore tonnages are not extracted from Hangar Flats until Year 6 and West End until Year 7, but waste rock mining starts several years before. To determine the correct size cone to guide phase design, multiple schedules were developed by scheduling the Yellow Pine phases followed by varying sized Hangar Flats and West End floating cones. Schedule results were then evaluated on a net present value basis. Table 15.6 summarizes the inputs that were used to evaluate the open pit sizing schedule options.

Table 15.6 Hangar Flats and West End Mine Schedule Inputs

Input	Value
Mining Cost	$1.73/st
Approximate capital cost to increase mining capacity	$2.00/st of annual production capacity
Block Value	Net of Process in $/st

For both Hangar Flats and West End, the $1,100/oz cones provided the best present value when evaluated on a time value of money basis and were chosen as the guidance cones for phase design.

15.5 GUIDANCE CONES AND ULTIMATE PIT DESIGNS

Floating cones or other computer generated pits do not consider phase access or bench working room and cannot be used for practical operations. The cones are used only as a guide for the design of operational mining phases. The following items were considered in phase design:

- Slope Angle Constraints: Strata provided inter-ramp and overall slope angle restrictions by sector for the three Mineral Reserve pits. The detailed slope constraints can be found in Section 16. Overall slope angle requirements are respected regardless of inter-ramp angles. While mining occurs on 20-ft benches,

benches are mined to a 40-ft double bench configuration at the pit wall to increase the catch benches to an appropriate width.

- Access: Access to every bench of every phase is incorporated into the pit design. For example, the highest road on the eastern wall of the Yellow Pine pit on Figure 15.5 is incorporated into the pit design to allow access to the West End pit later in the mine life.
- Haul Road Pit Exits: Pit exit locations are chosen to have the haul road exit the pit at the most beneficial location for the haulage of ore to the crusher and waste to the appropriate storage location.
- Realistic Mining Geometries: Computer generated pits have irregular pit walls that will be operationally difficult to mine. Designing pits removes these irregularities and smooths out pit walls.

By incorporating haul roads into the pit design and smoothing irregularities in the pit walls, additional waste rock is often incurred by expanding the pit in the upper benches and some ore that was in the bottom benches of the floating cones is left behind and not mined. For this reason, designed phases often have a higher stripping ratio than optimized cones.

Open pit design criteria are provided in Table 15.7.

Table 15.7 Design Parameters for Mine Pits

Design Parameter	Parameters Value
Haul Road Width Including Ditches and Berms	102 ft
Maximum Haul Road Grade	10%
Bench Height for Mining	20 ft
Face Angle of Benches	64^{o} (Double Benched)
Overall Slope Angles Used	Variable between 39^{o}- 47^{o}
Inter-ramp Slope Angles Used	Variable between 45^{o}- 49^{o}

The design cones for Yellow Pine, Hangar Flats and West End are illustrated on Figure 15.4, Figure 15.6, and Figure 15.8, respectively. The ultimate pits for Yellow Pine, Hangar Flats and West End are shown at the same scale on Figure 15.5, Figure 15.7, and Figure 15.9, respectively, and presented adjacent for comparative purposes.

Figure 15.4 Yellow Pine Cone Floated at $800/oz Gold Price

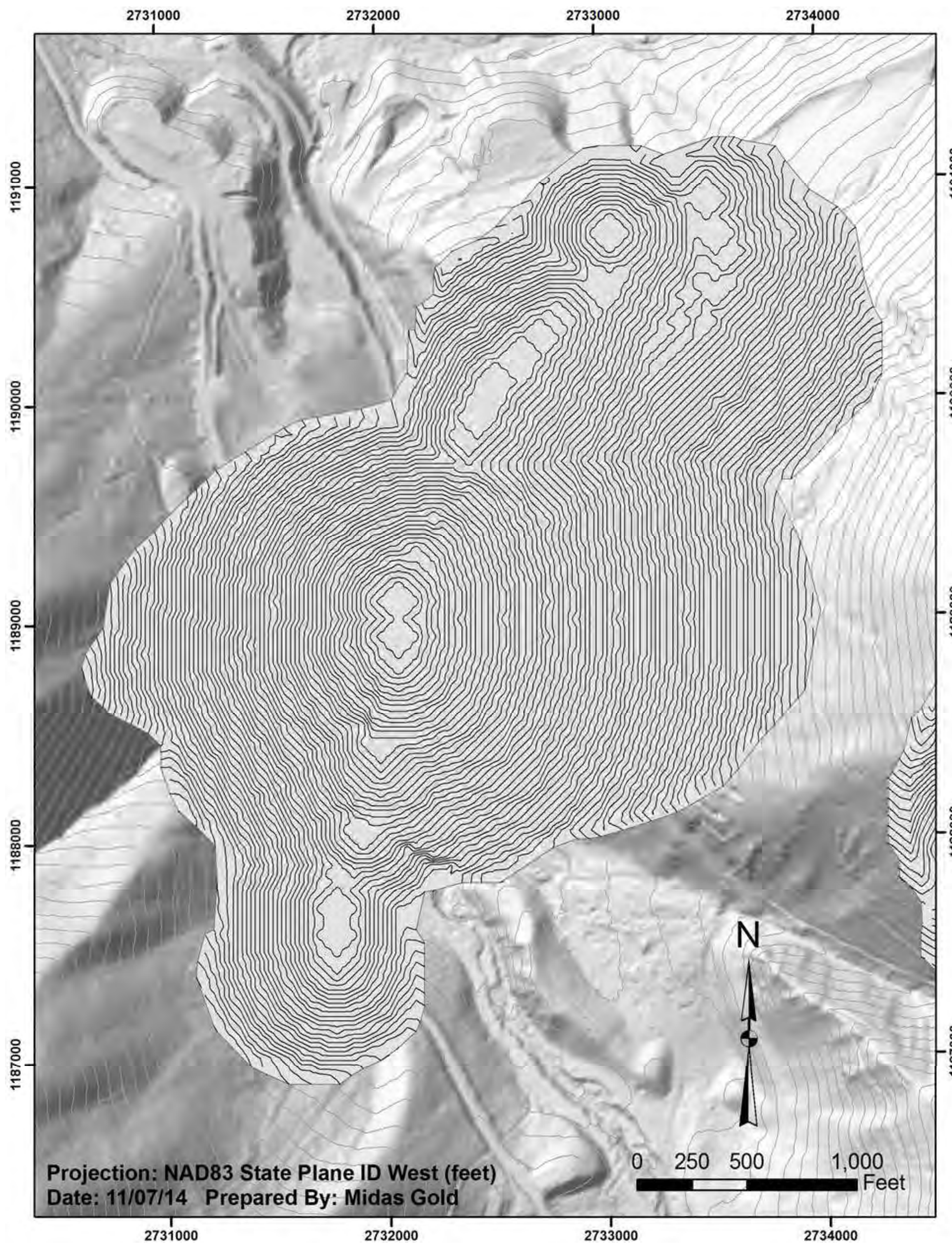


Figure 15.5 Yellow Pine Ultimate Pit

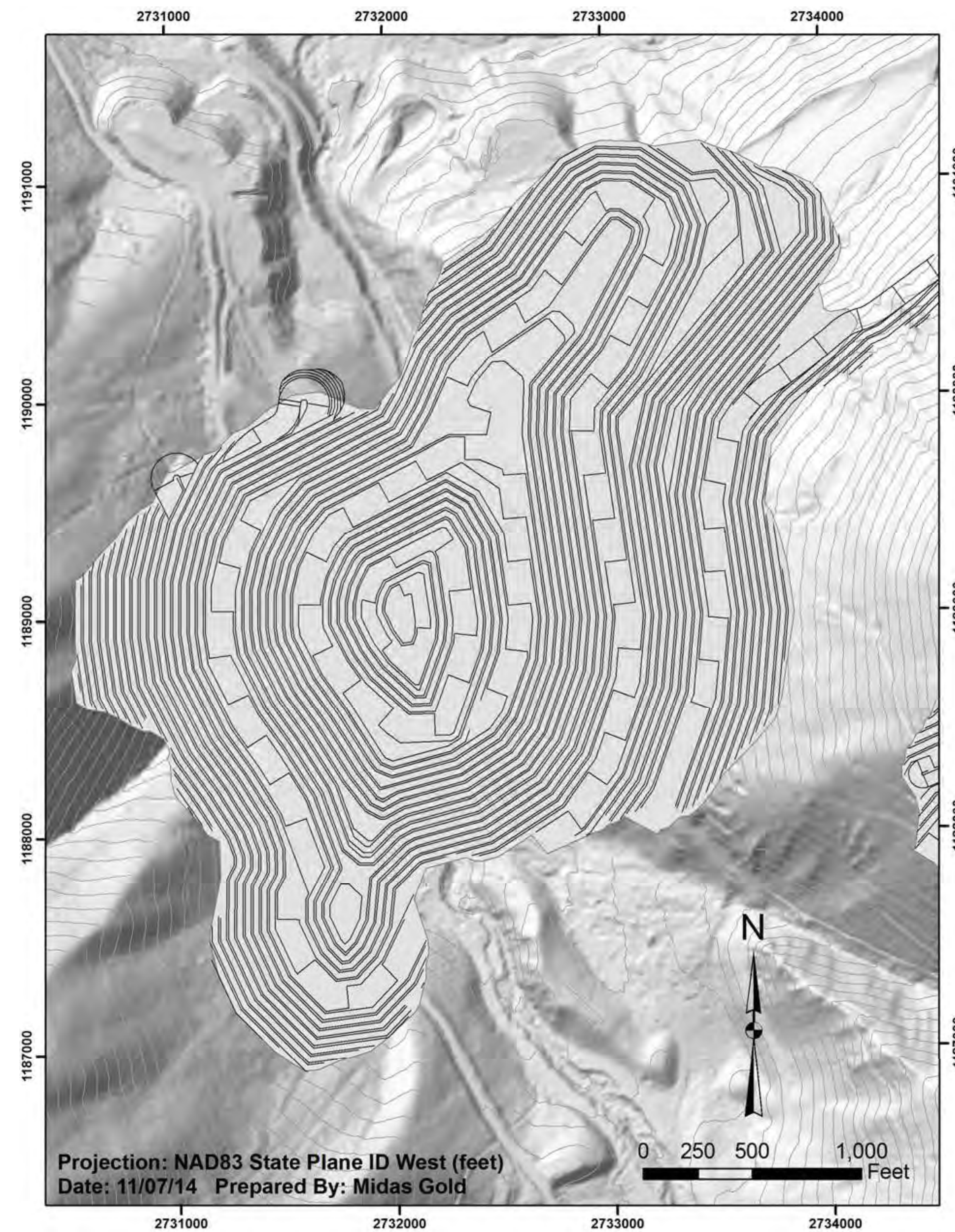


Figure 15.6 Hangar Flats Cone Floated at $1,100/oz Gold Price

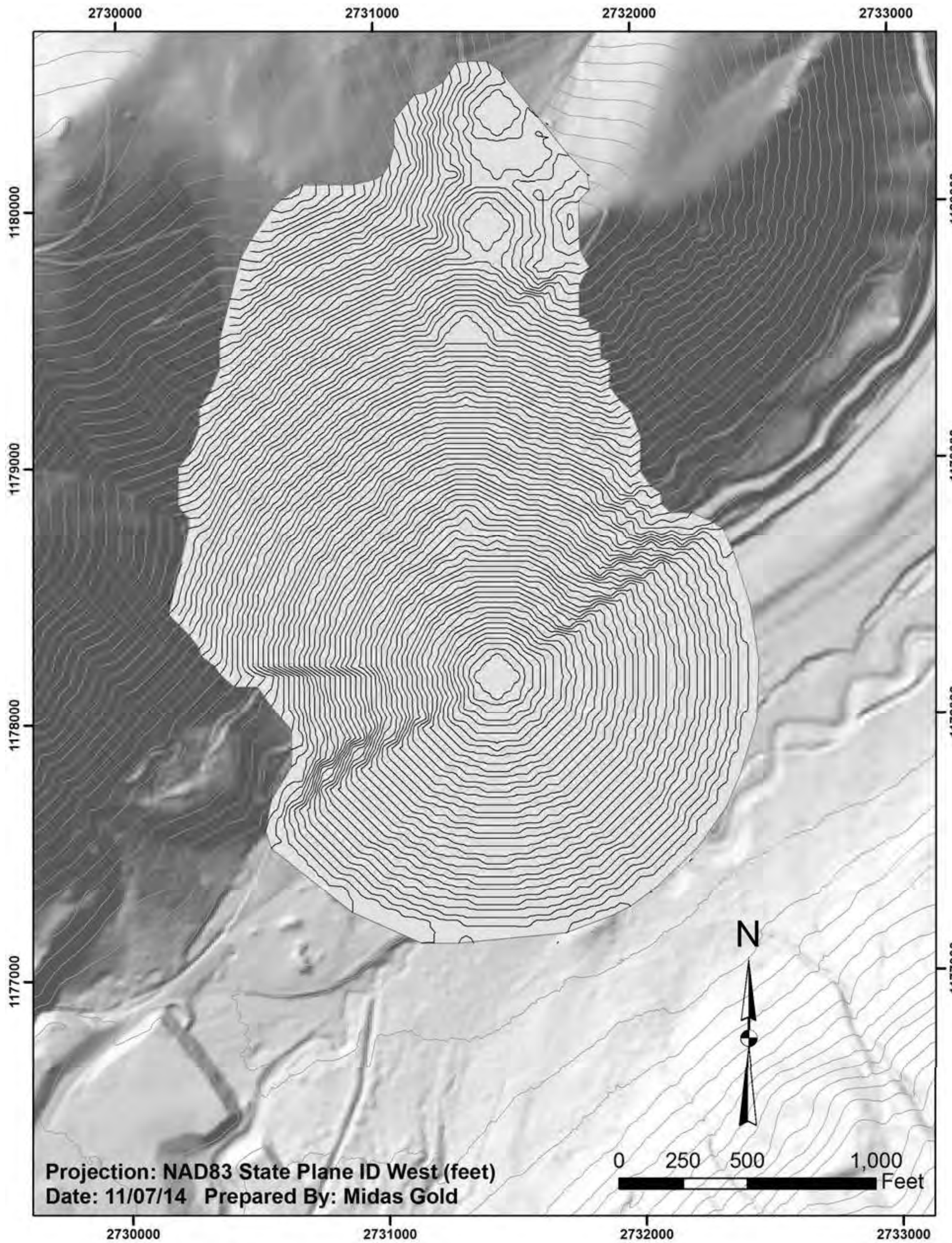


Figure 15.7 Hangar Flats Ultimate Pit

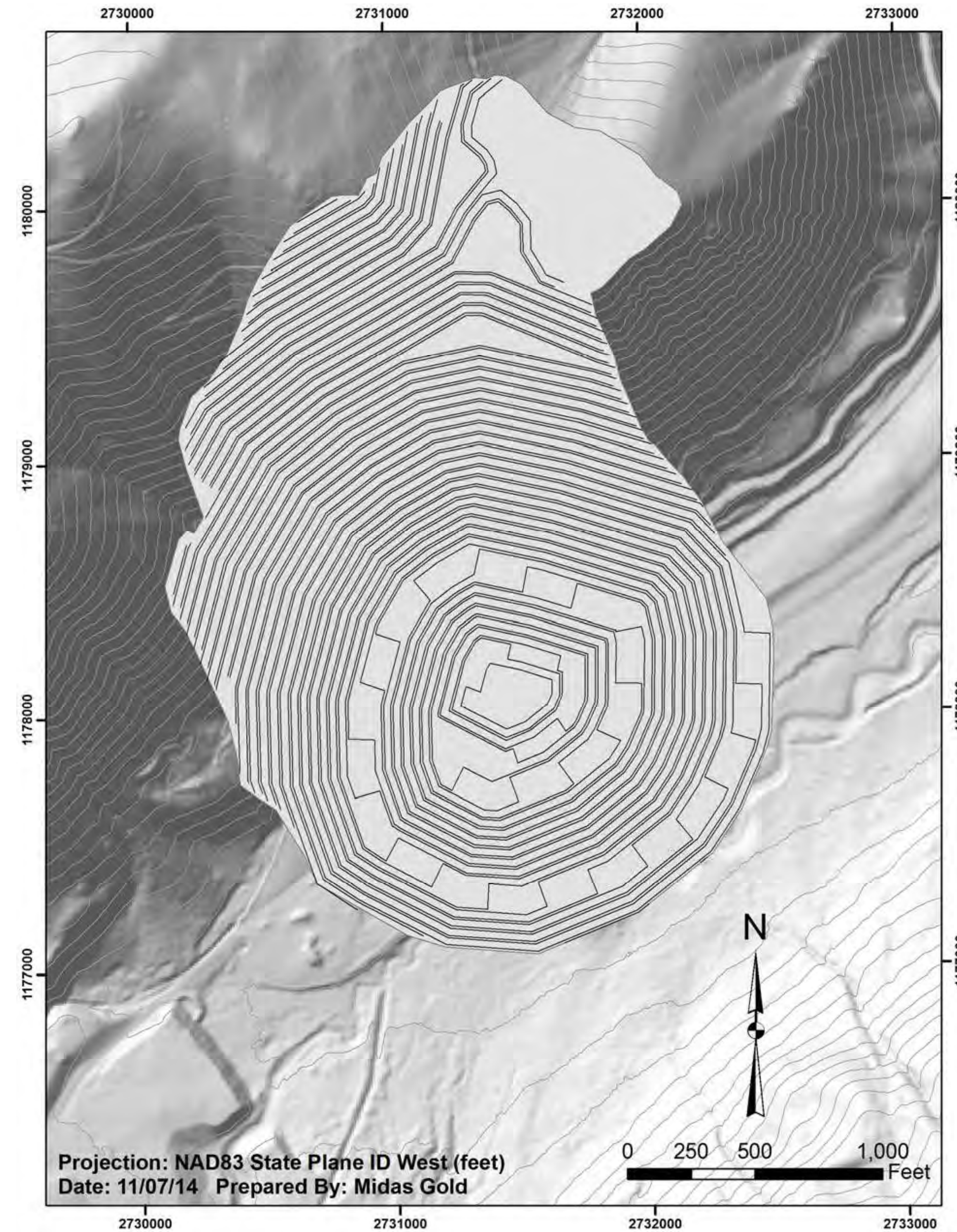


Figure 15.8 West End Cone Floated at $1,100/oz Gold Price

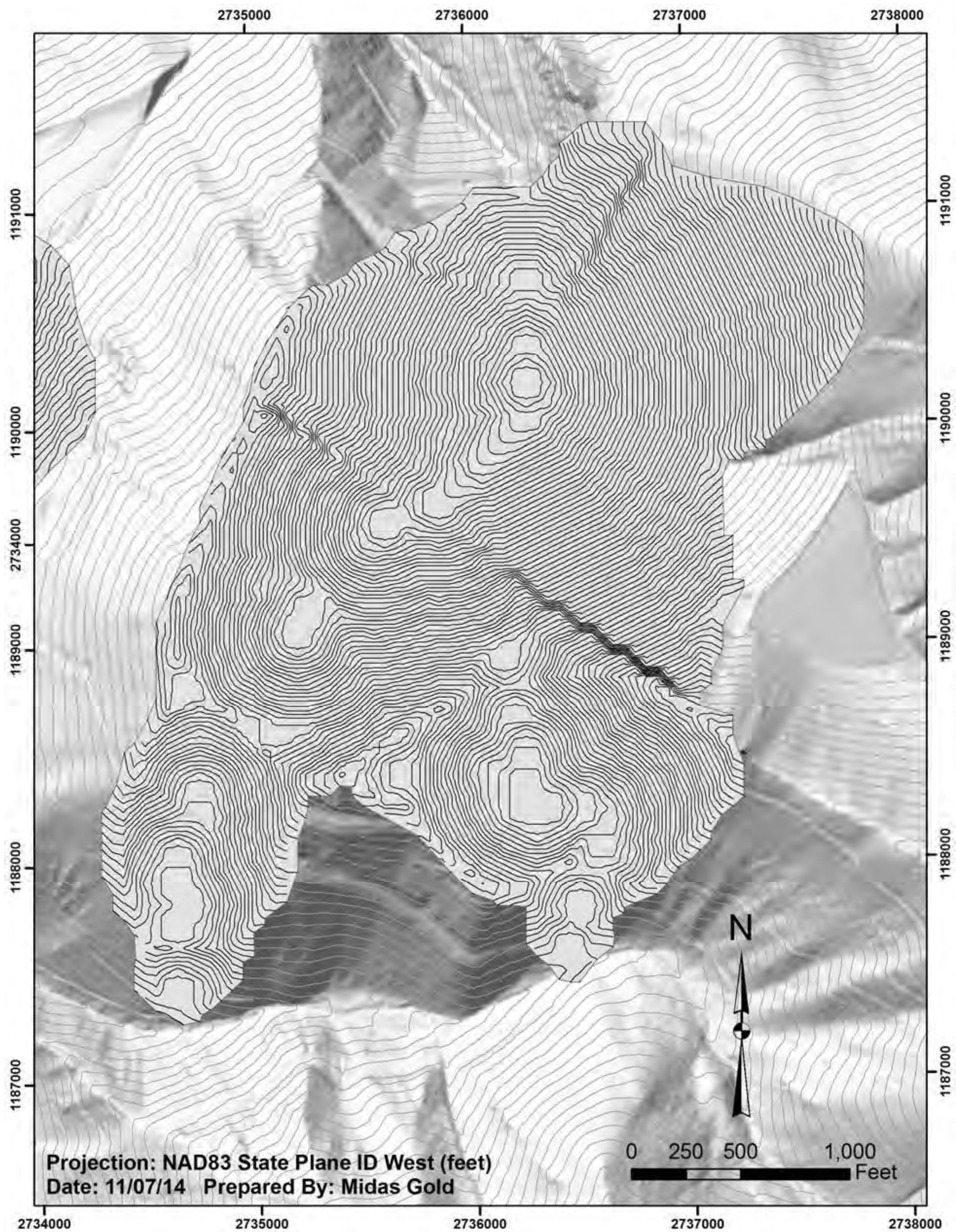
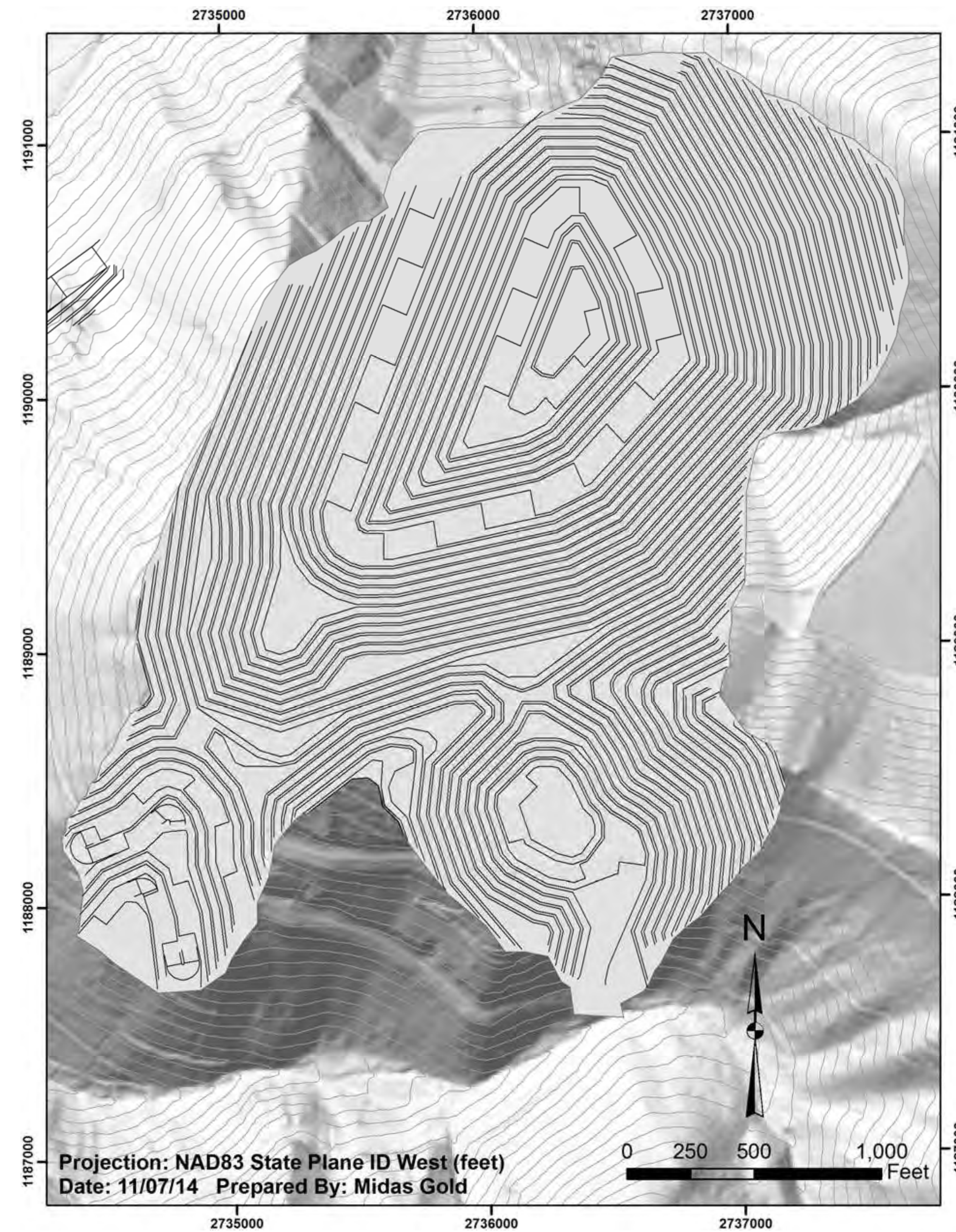


Figure 15.9 West End Ultimate Pit



15.6 HISTORIC TAILINGS

Southwest of the Hangar Flats open pit, and within the planned waste rock storage facility footprint, lies the Historic Tailings impoundment. Metallurgical test results show that the contained gold values in the Historic Tailings produces an economic benefit when fed to the process plant concurrent to primary ores; consequently, the Historic Tailings are planned to be mined and processed through the mill and are included in the Mineral Reserve. The impoundment contains 3,164 kst of Historic Tailings of which 3,001 kst are Indicated Mineral Resources.

15.7 PRE-FEASIBILITY UPDATED COSTS AND RECOVERIES

Final PFS recoveries and processing costs became available towards the end of the PFS work. The block model values were updated with these revised inputs for mine planning purposes. The final PFS inputs used to calculate block NPR values can be found in accompanying sections of this Report. Metal recoveries can be found in Section 13. Transport costs and smelter terms can be found in Section 19. Processing costs used were not final at the time that the Mineral Reserve was defined and those used are provided in Table 15.8; even though the processing costs were revised after the Mineral Reserve was defined, the change in processing costs would not produce a material change in the Mineral Reserve. The final processing costs are incorporated into the Project cash flow included in Section 22. Metal Prices of $1,350/oz. gold, $22.50/oz. silver, and $4.50/lb antimony were used. The logic used to assign process types to each block is similar to the logic that was used for assigning process types for the cone inputs. Updates to processing flow sheets reduced the ore types in West End from three to two: oxide material and mixed sulfide material. The NPR equations are provided in Table 15.9. Ore type is determined by the process that produces the greatest NPR.

IMC conducted a sensitivity check on the impact of incorporating the updated inputs and determined that the existing phase designs are acceptable for use in prefeasibility-level mine planning and no re-design was necessary.

Table 15.8 PFS Processing and G&A Costs by Mineral Resource Type

Mineral Resource Type	Unit Cost
High Antimony Sulfide	$17.00/st
Low Antimony Sulfide	$15.00/st
Oxide	$9.07/st
All Ore Types G&A	$3.40/st

Table 15.9 PFS Logic for Calculating Block Net of Process Values

For Yellow Pine and Hangar Flats Greatest Value of:	High Antimony Mineral Resource Net of Process	Au Grade x (Au Float Rec. + Au Tail Leach Rec.) x POX rec. x % Dore Payable x (Au Price – Au Refining Charge) x (1 – Royalty) + Ag Grade x (Au Float Rec. + Au Tail Leach Rec.) x POX rec. x % Dore Payable x (Ag Price – Ag Refining Charge) + If Au in Sb Concentrate > 0.146 oz/st: Au Grade x Au loss to Sb Flotation x (Au Smelter Payable – Royalty) x Au Price + If Ag in Sb Concentrate > 8.75 oz/st: Ag Grade x Ag Rec. to Sb Flotation x Ag Smelter Payable x Ag Price + Sb Grade / 100 x Sb Flotation Recovery x Sb Smelter Payable x Sb Price x 2000 - TCRC x Sb Grade x Sb Flotation Recovery / Sb Concentrate Grade - High Antimony Processing Cost
	Low Antimony Mineral Resource Net of Process	Au Grade x (Au Float Rec. + Au Tail Leach Rec.) x POX rec. x % Dore Payable x (Au Price – Au Refining Charge) x (1 – Royalty) + Ag Grade x (Au Float Rec. + Au Tail Leach Rec.) x POX rec. x % Dore Payable x (Ag Price – Ag Refining Charge) - Low Antimony Processing Cost
	Waste Net of Process	Zero
For West End Greatest Value of:	Low Antimony Mineral Resource Net of Process	Au Grade x (Au Float Rec. + Au Tail Leach Rec.) x POX rec. x % Dore Payable x (Au Price – Au Refining Charge) x (1 – Royalty) + Ag Grade x (Au Float Rec. + Au Tail Leach Rec.) x POX rec. x % Dore Payable x (Ag Price – Ag Refining Charge) - Low Antimony Processing Cost
	Oxide Mineral Resource Net of Process	Au Grade x % Dore Payable x Oxide Au Recovery x (Au Price – Au Ref Charge) x (1- Royalty) + Ag Grade x % Dore Payable x Oxide Ag Recovery x (Ag Price – Ag Ref Charge) - Oxide Processing Cost
	Waste Net of Process	Zero

Notes:
(1) Au and Ag grades are in oz/st.
(2) % Dore payable is a decimal value.
(3) Royalty is a decimal value.
(4) Au and Ag price is in $/oz.
(5) Sb price is in $/lb.
(6) Au and Ag refining charges are in $/oz.
(7) Processing costs are in $/st.
(8) rec. is recovery in decimal form.

15.8 MINERAL RESERVE ESTIMATE

The designation of Indicated Mineral Resources to the Mineral Reserve category (there are no Measured Mineral Resources) is based on the final PFS process plant metallurgical recoveries, processing costs, and smelter terms. The Mineral Reserve is the sum of the Probable material (there is no Proven material) that is scheduled to be processed in the mine plan that is presented in detail in Section 16. The cutoff grade for material sent to processing ranges from $0.001/st - $8.00/st Net of Process Revenue.

The processing costs used for mine planning ranged from $9.07/st for oxides to $17.00/st for high antimony sulfides with an additional $3.40/st of ore for general and administrative expenses. Therefore, the NSR equivalent of the cutoff grade range is: $12.47/st – $20.40/st Net of Smelter Return. The Mineral Reserves are summarized in Table 15.10 in both imperial and metric units.

Table 15.10: Stibnite Gold Project Probable Mineral Reserves Summary

Deposit	Tonnage	Average Grade			Total Contained Metal		
		Gold	Antimony	Silver	Gold	Antimony	Silver
Imperial Units	(kst)	(oz/st)	(%)	(oz/st)	(koz)	(klbs)	(koz)
Yellow Pine	43,985	0.057	0.098	0.090	2,521	86,376	3,973
Hangar Flats	15,430	0.045	0.132	0.086	690	40,757	1,327
West End	35,650	0.035	0.000	0.040	1,265	-	1,410
Historic Tailings	3,001	0.034	0.165	0.084	102	9,903	252
Total Probable Mineral Reserve(1)	98,066	0.047	0.070	0.071	4,579	137,037	6,962
Metric Units	(kt)	(g/t)	(%)	(g/t)	(t)	(t)	(t)
Yellow Pine	39,903	1.97	0.098	3.10	78.4	39,179	123.6
Hangar Flats	13,998	1.53	0.132	2.95	21.5	18,487	41.3
West End	32,341	1.22	0.000	1.36	39.3	-	43.9
Historic Tailings	2,722	1.17	0.165	2.88	3.2	4,492	7.8
Total Probable Mineral Reserve(1)	88,964	1.60	0.070	2.43	142.4	62,159	216.5

Notes:
(1) *Metal prices used for Mineral Reserves: $1350/oz Au, $22.50/oz Ag, $4.50/lb Sb.*
(2) *Block MUST be economical based on gold value only in order to be included as ore in Mineral Reserve.*
(3) *Numbers may not add exactly due to rounding.*

Illustrations of the Yellow Pine, Hangar Flats and West End Mineral Resources and mineralized material that are not part of the reported Mineral Reserves are presented as Figure 15.10, Figure 15.11, and Figure 15.12, respectively.

Figure 15.10: YP Mineral Reserves and Mineralized Material in Plan, Section, and 3D Perspective Views

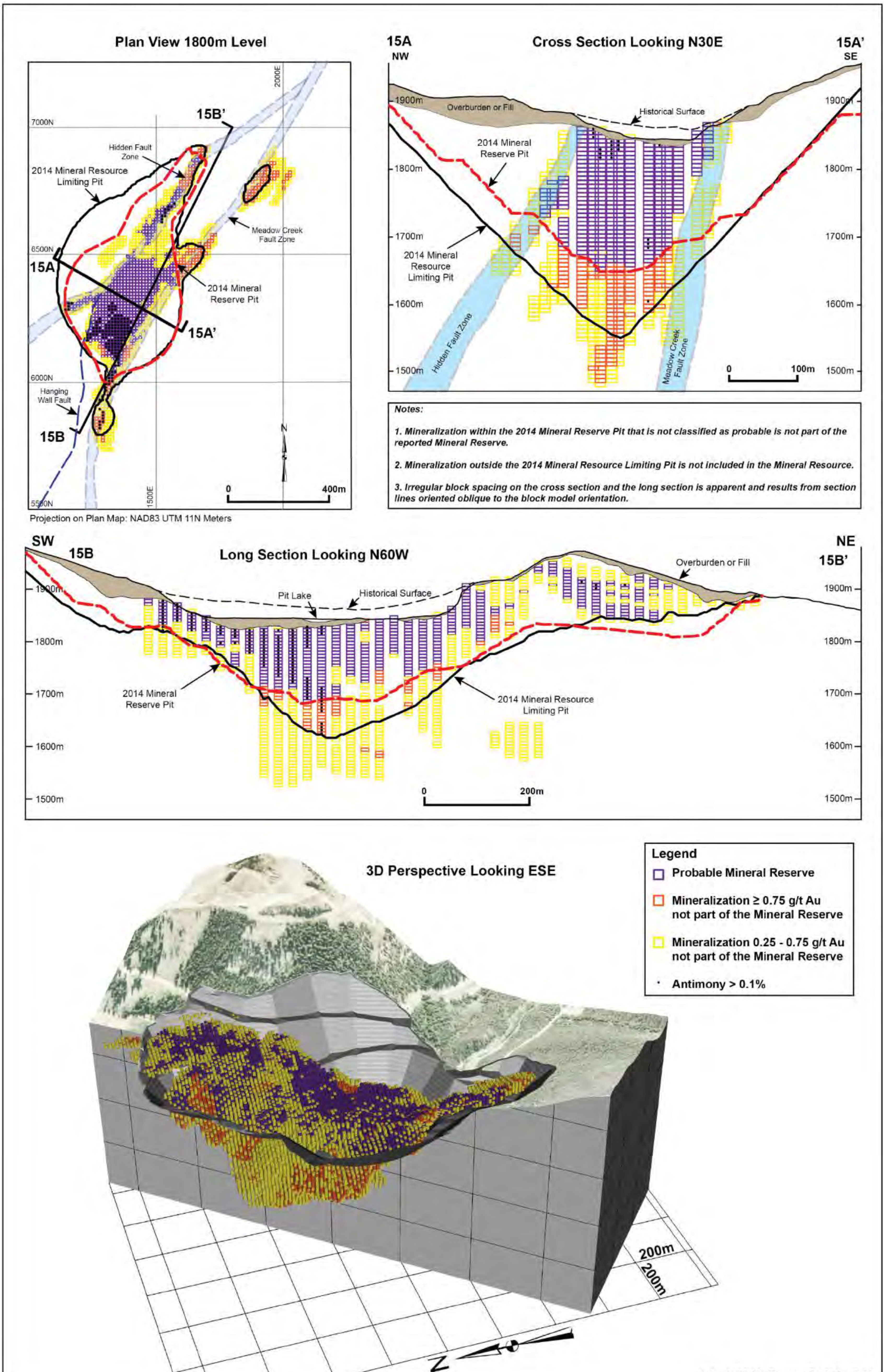


Figure 15.11: HF Mineral Reserves and Mineralized Material in Plan, Section, and 3D Perspective Views

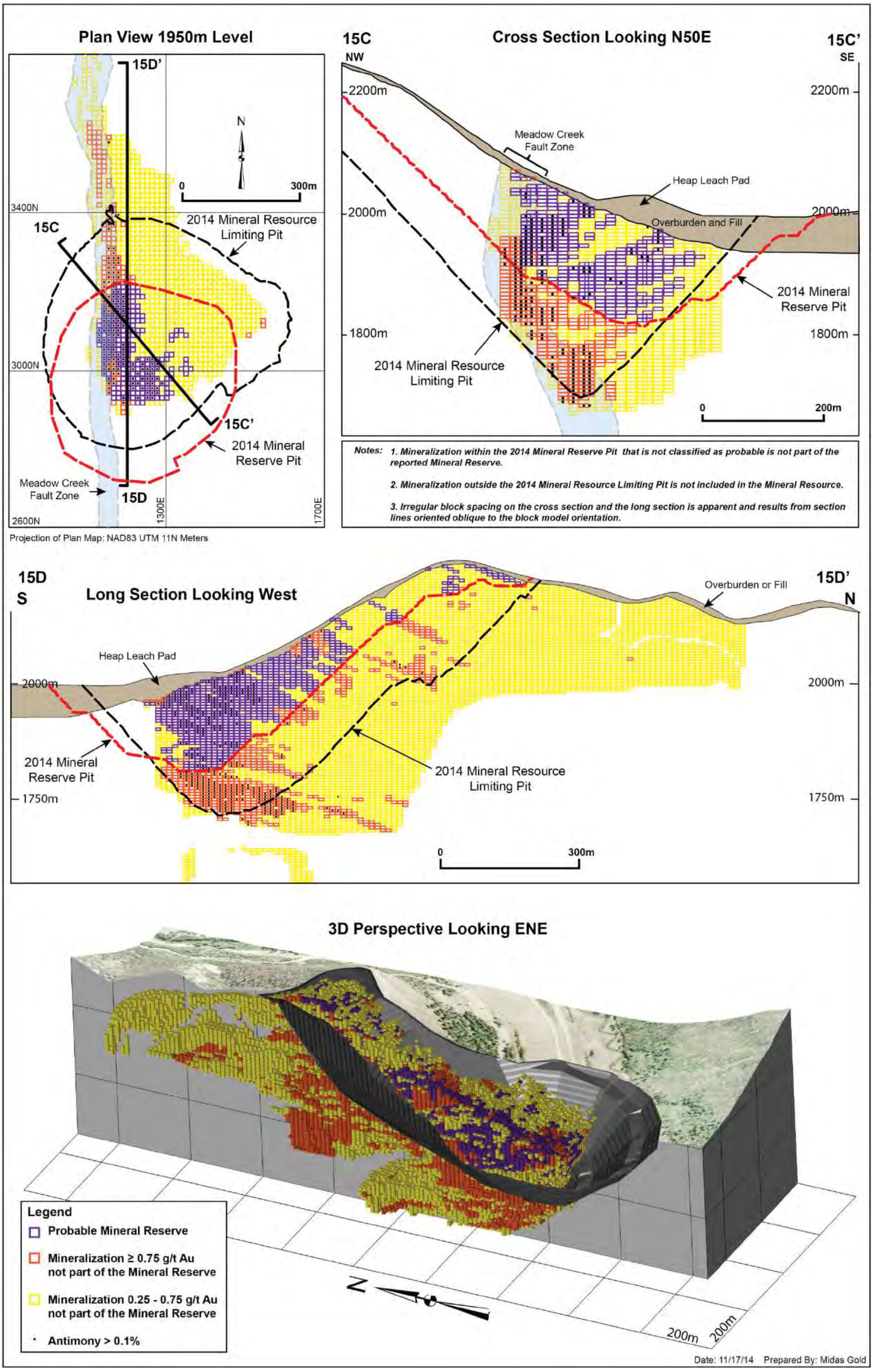
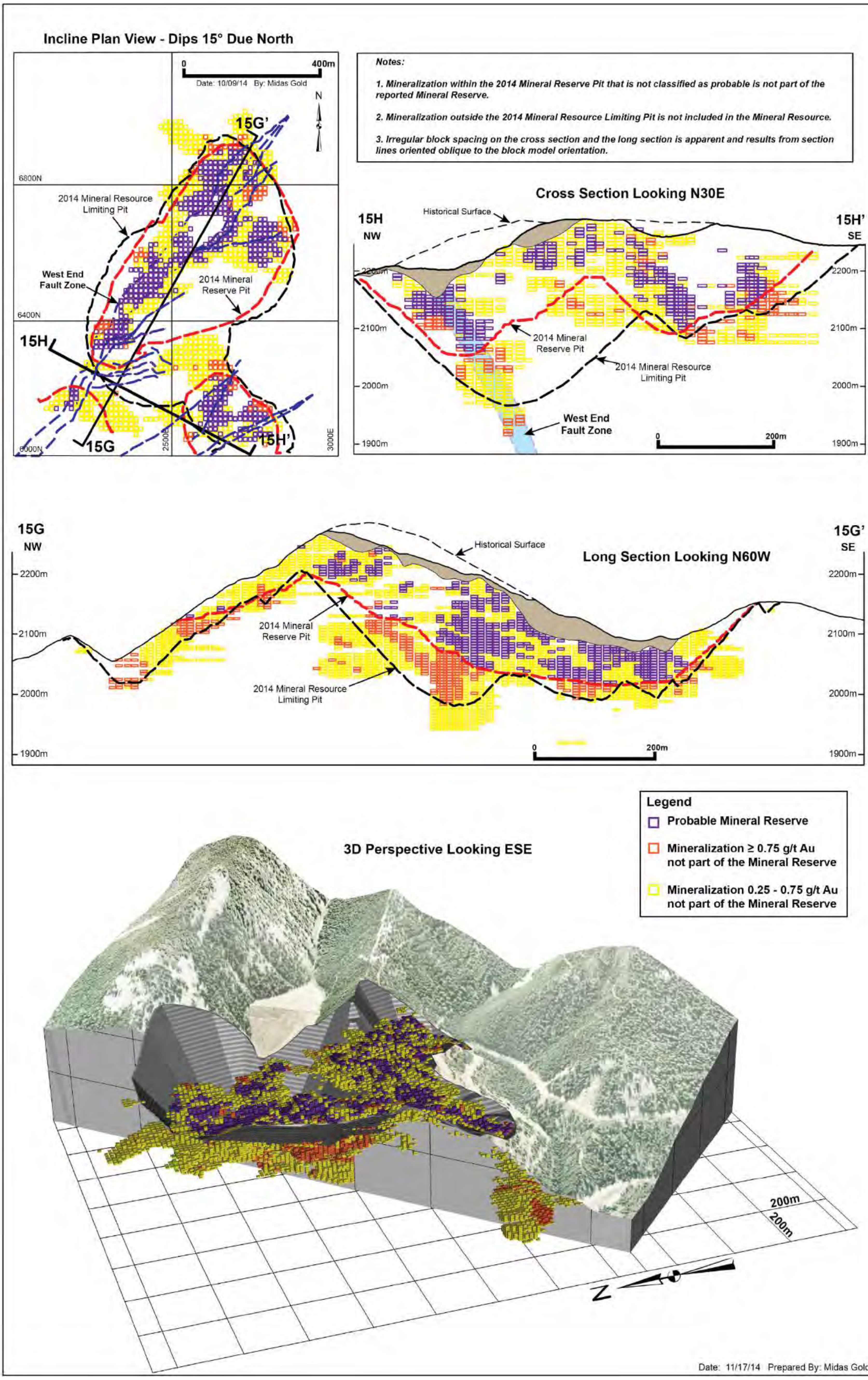


Figure 15.12: WE Mineral Reserves and Mineralized Material in Plan, Section, and 3D Perspective Views



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16 MINING METHODS

16.1 INTRODUCTION

The Stibnite Gold Project PFS mine plan was developed using conventional open pit hard rock mining methods. The mining operation is planned to deliver 8.05 million tons of ROM material to the primary crusher per year (nominally 22,050 short tons per day). Both oxide and sulfide mineralized material would be sent to the crusher; the oxide material would be vat-leached while the sulfide material would be processed via up to two sequential flotation circuits to produce two concentrates: (1) if sufficient antimony grade is present, an antimony concentrate that would be filtered, trucked off-site and sold; and, for all sulfide material, (2) an auriferous sulfide concentrate that would be oxidized onsite via pressure oxidation, then processed via agitated leach, carbon stripping and refining to produce a gold- and silver-rich doré.

The mine plan developed for the Project incorporates the mining of three primary mineral deposits – Yellow Pine, Hangar Flats, and West End – and re-mining and re-processing of the Historic Tailings. Mineral Reserves from the three open pits would be sent to a centrally located primary crusher, while the Historic Tailings would be mined with an excavator and trucks then hydraulically transferred from an adjacent pulping facility to the process plant grinding circuit. Waste rock would be sent to four distinct destinations: the TSF embankment, the Main WRSF, the West End WRSF, and to the mined-out Yellow Pine open pit. The general sequence of mining is the Yellow Pine deposit first, Hangar Flats second, and the West End deposit third, while Historic Tailings would overlap with the Yellow Pine deposit processing. The mining sequence is influenced by the need to backfill the Yellow Pine open pit to restore the original gradient of the EFSFSR; this order also generally follows a sequence of mining highest to lowest grade, which is also preferred.

The Historic Tailings are situated within the footprint of the proposed Main WRSF, and would be re-mined and reprocessed during the first four years of the mine schedule to provide adequate terrain for the WRSF. As the tailings material is currently at a size of 97% passing #40 mesh, it is not anticipated to overwhelm the process plant through the additional throughput. The tailings are currently overlain with 5,752 kst of neutralized spent heap leach ore (commonly referred to as the SODA) that must be removed before the Historic Tailings can be mined. The SODA material is planned to be used in the construction of the TSF starter dam during pre-production.

A summary of the ore tonnage by process type and waste tonnage from each of the primary deposits and the Historic Tailings is provided in Table 16.1. These tonnages correspond with the mine schedule provided in Table 16.7.

Table 16.1: Summary of Mine Plan Ore Type and Tonnage and Waste by Deposit

Resource	Ore Type	Ore Tons (kst)	Gold Grade (oz/st)	Silver Grade (oz/st)	Antimony Grade (%)	Waste Tons (kst)	Strip Ratio (st:st)
Yellow Pine	High Sb	6,750	0.065	0.210	0.593	124,304	2.8:1
	Low Sb	37,235	0.056	0.069	0.009		
Hangar Flats	High Sb	4,284	0.056	0.166	0.425	86,696	5.6:1
	Low Sb	11,146	0.040	0.055	0.019		
West End	Oxide	10,736	0.022	0.029	-	129,995	3.6:1
	Low Sb	24,914	0.041	0.044	-		
Historical Tailings	Low Sb	3,001	0.034	0.084	0.165	5,915	2.0:1
Totals / Averages		98,066	0.047	0.071	0.070	346,910	3.5:1

In addition to mine sequencing constraints, the PFS mine schedule was developed considering requirements of the processing plant. These are: maintenance down time and sulfur content restrictions for the feed to the POX circuit. Oxide material would be stockpiled adjacent to the primary crusher and processed during planned POX circuit maintenance to address the expected additional availability of the comminution and leach circuits over the POX circuit. To address the sulfur feed limitation, the mining schedule was developed by considering the maximum nominal sulfur levels that the POX circuit could handle in a given quarter. The peak sulfur levels generally occur in the initial 2 years of mine operations and generally correspond to the highest gold grades in the Yellow Pine deposit; after that period, the process plant would no longer be constrained by high sulfur levels.

The Independent Mining Consultants, Inc. (IMC) mine planning team applied the following steps to develop the Stibnite Gold Project PFS mine plan:

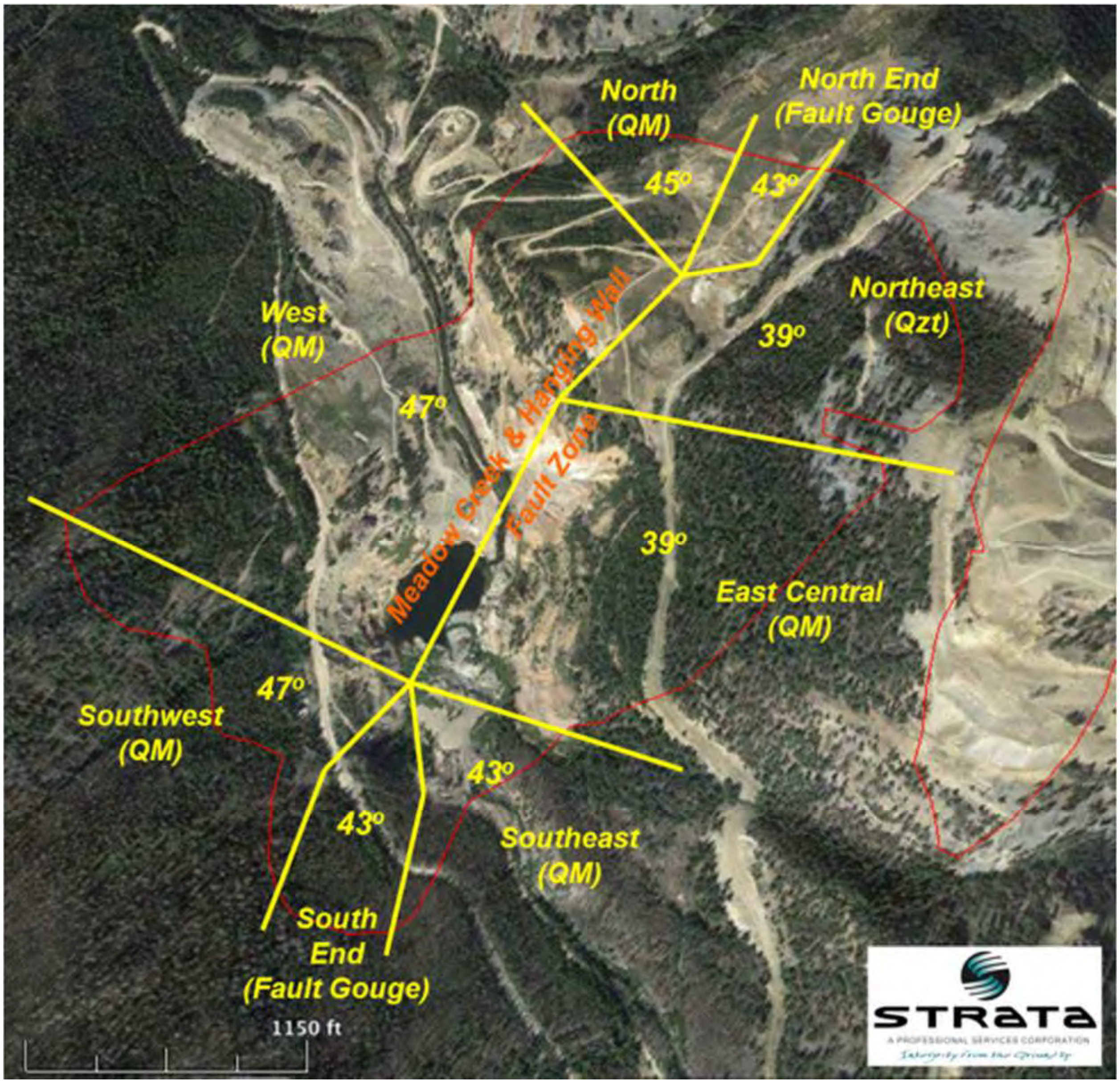
1) floating cone guidance for phase design;
2) phase designs;
3) mine production schedule;
4) waste rock storage design and waste rock allocation;
5) haul road design;
6) time sequence mine and dump drawings; and
7) equipment and manpower requirements.

Additional details associated with the preceding steps are described in the following subsections.

16.2 GEOTECHNICAL CONSIDERATIONS

Figure 16.1, Figure 16.2, and Figure 16.3 show the overall slope angles and sectors provided by the Project geotechnical consultant Strata, A Professional Services Corporation (Strata) for the Yellow Pine, Hangar Flats and West End open pits, respectively. The red lines on the figures are approximate ultimate open pit crests. Table 16.2, Table 16.3 and Table 16.4 provide the inter-ramp slope angles used in the Yellow Pine, Hangar Flats, and West End open pit designs provided by Strata, respectively. Overall slope angles were respected regardless of inter-ramp angles.

Figure 16.1: Overall Slope Angles Used at Yellow Pine Open Pit Sectors

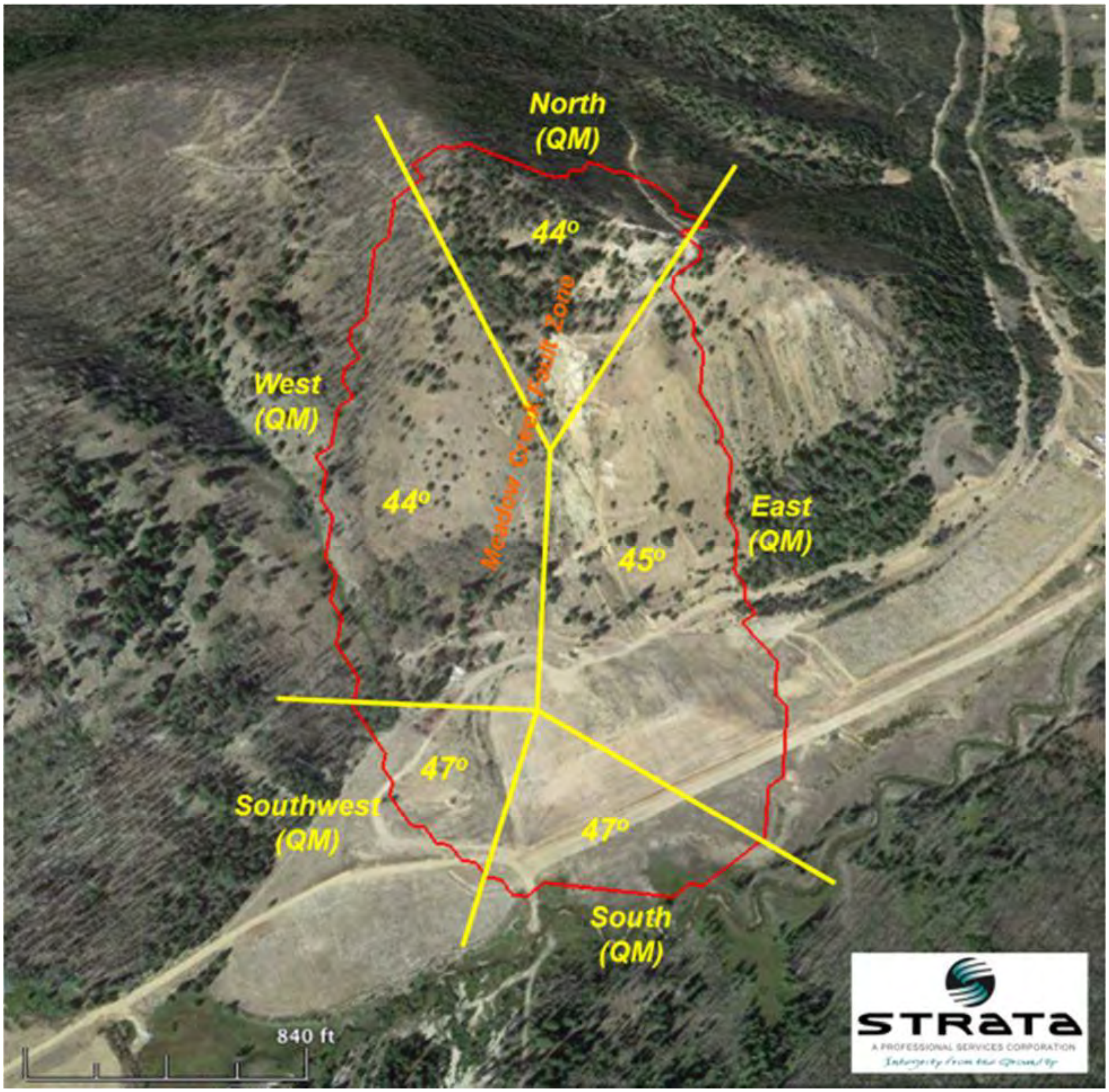


From Strata (figure oriented as north up).

Table 16.2: Yellow Pine Open Pit Inter-Ramp Slope Angles

Sector	North	West	Southwest	Southeast	East Central	Northeast
Average Slope Dip Direction	175° - 185°	147°	43° - 65°	293° - 303°	317°	230° - 250°
Inter-Ramp Slope Angle	47°	47°	48°	47°	47°	49°
Note: Open pit inter-ramp slope angles estimated by Strata.						

Figure 16.2: Overall Slope Angles Used at Hangar Flats Open Pit Sectors

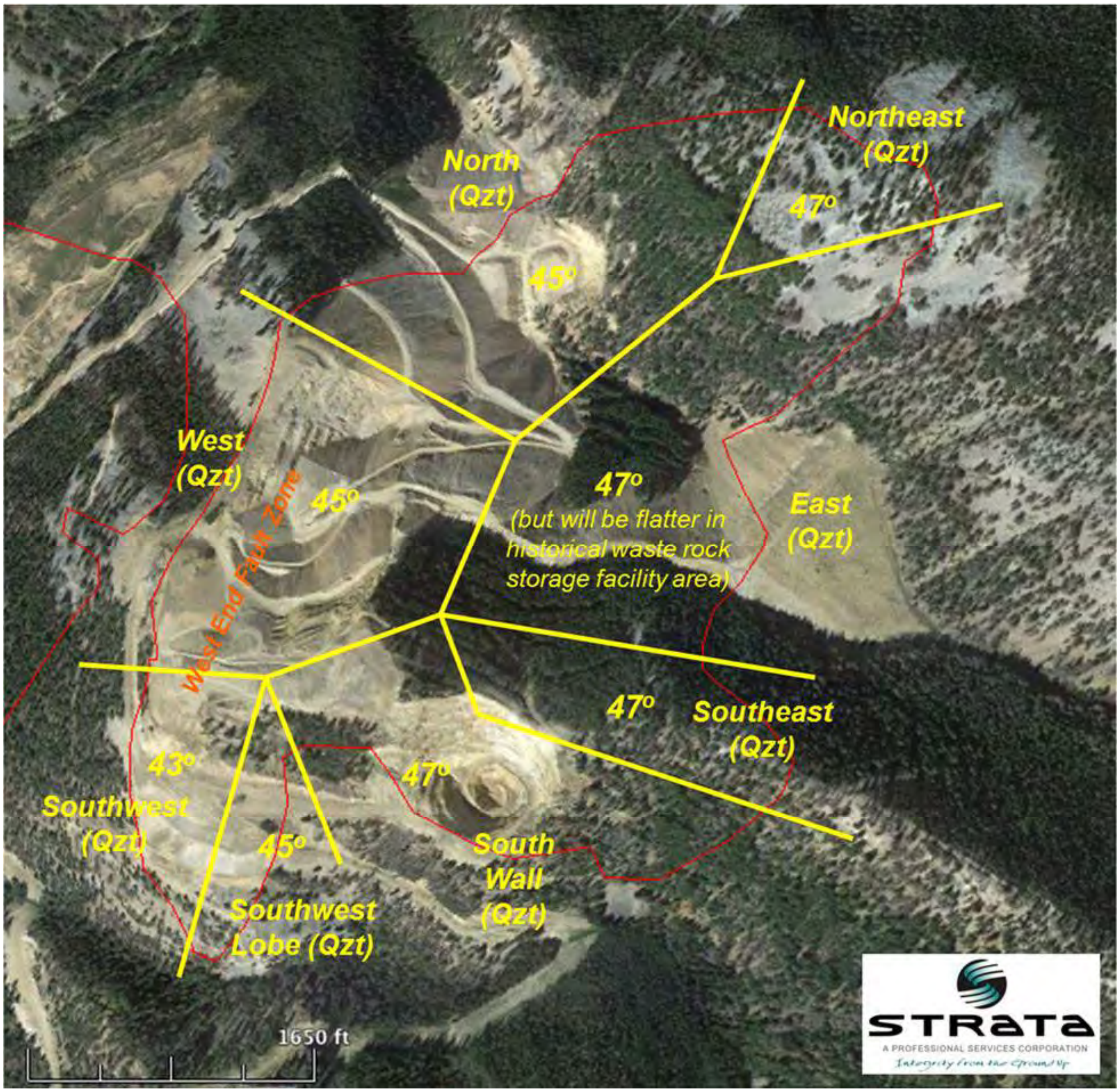


From Strata (figure oriented as north up).

Table 16.3: Hangar Flats Open Pit Inter-Ramp Slope Angles

Sector	North	West	Southwest	South	East
Average Slope Dip Direction	190°	96°	53°	345° - 360°	263°
Inter-Ramp Slope Angle	45°	47°	47°	47°	45°
Note: Open pit inter-ramp slope angles estimated by Strata.					

Figure 16.3: Overall Slope Angles Used at West End Open Pit Sectors



From Strata (figure oriented as north up).

Table 16.4: West End Open Pit Inter-Ramp Slope Angles

Sector	Northeast	Northeast	West	Southwest	Southwest Lobe	South Wall	Southeast	East
Average Slope Dip Direction	207°	140° – 160°	112°	73°	290°	0° – 18°	232°	298°
Inter-Ramp Slope Angle	47°	45°	45°	47°	47°	47°	47°	47°

Note: Open pit inter-ramp slope angles estimated by Strata.

16.3 PHASE DESIGN

The final PFS phase designs were guided by the floating cone pit shells that were described in Section 15. Phases are designed to even out waste rock stripping over the mine life and to move higher-grade ore forward in the mine schedule. The culmination of the phase designs results in the ultimate pits that were presented in Section 15. Phase designs include all internal access roads and assure proper operating requirements for mining equipment.

A total of three phases were designed to achieve the ultimate Yellow Pine open pit; an initial phase followed by an eastern extension, then a western extension. The initial phase of Yellow Pine requires 3,734 thousand tons (kst) of waste rock stripping to expose sufficient ore for production at the planned throughput rate. Additional waste rock movement is shown in pre-production for TSF starter dam construction requirements. The second phase of Yellow Pine mines east of the initial phase because the stripping ratio is lower on this side of the open pit and consequently gold ounces are lower cost to the east side than to the west.

The Hangar Flats open pit is planned to be mined in a single phase with a small waste rock phase that would be mined in pre-production to provide material for the tailings dam construction. Hangar Flats is a single phase because the terrain of the northwestern high wall prevents access to a second pushback in that direction.

The West End open pit is planned to be mined in three phases; the second phase expanding in every direction except for the eastern wall. An initial oxide phase was designed within the ultimate West End open pit to accommodate process plant oxide feed material requirements during the first five years of mine life.

Generally, the three Yellow Pine open pit phases are mined first in the mine schedule, followed by Hangar Flats then the two West End phases. The parameters for the mine phase designs are summarized in Table 16.5.

Table 16.5: Design Parameters for Mine Phases

Design Parameter	Parameters Value
Haul Road Width Including Ditches and Berms	102 feet
Maximum Haul Road Grade	10%
Bench Height for Mining	20 feet
Face Angle of Benches	64^{o} (Double Benched)
Overall Slope Angles Used	Variable between 39^{o}- 47^{o}
Inter-ramp Slope Angles Used	Variable between 45^{o}- 49^{o}

The material mined from each phase on an annual basis is provided on Figure 16.4 and Figure 16.5. Open pit progression (at the end of preproduction and by year thereafter) as well as waste rock storage facility and haul road progression can be seen on Figure 16.12 through Figure 16.24, inclusively.

Figure 16.4: Ore Mining Schedule by Deposit and Phase

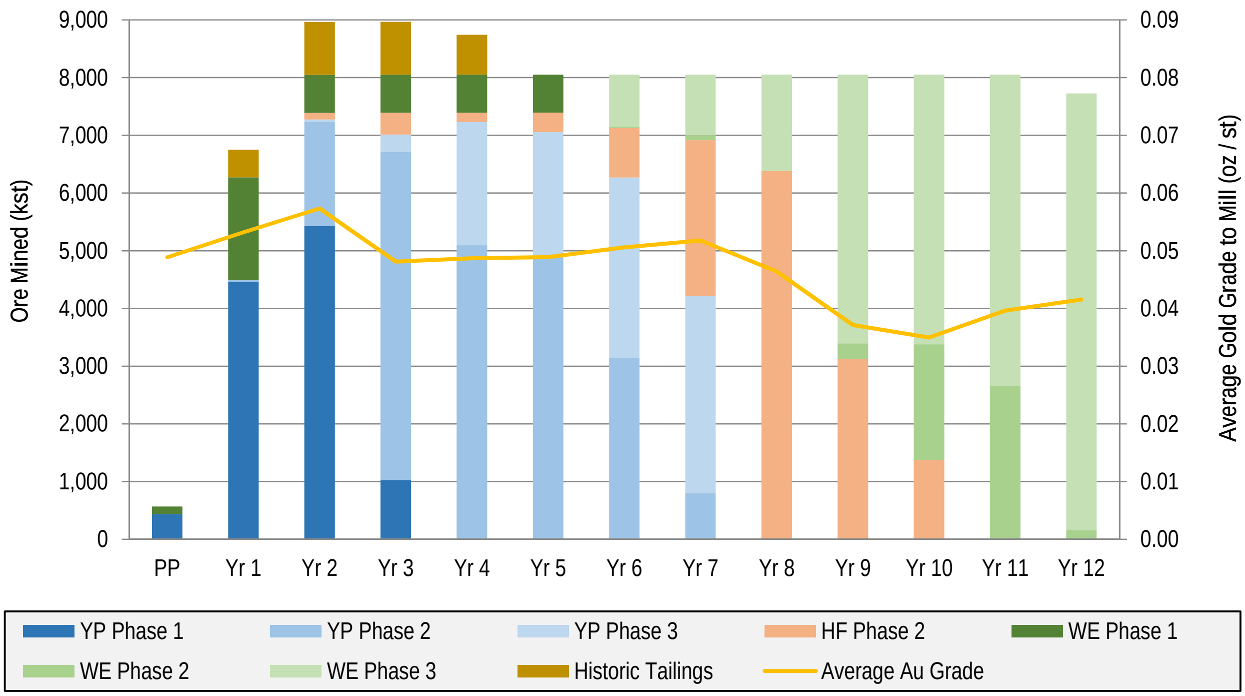
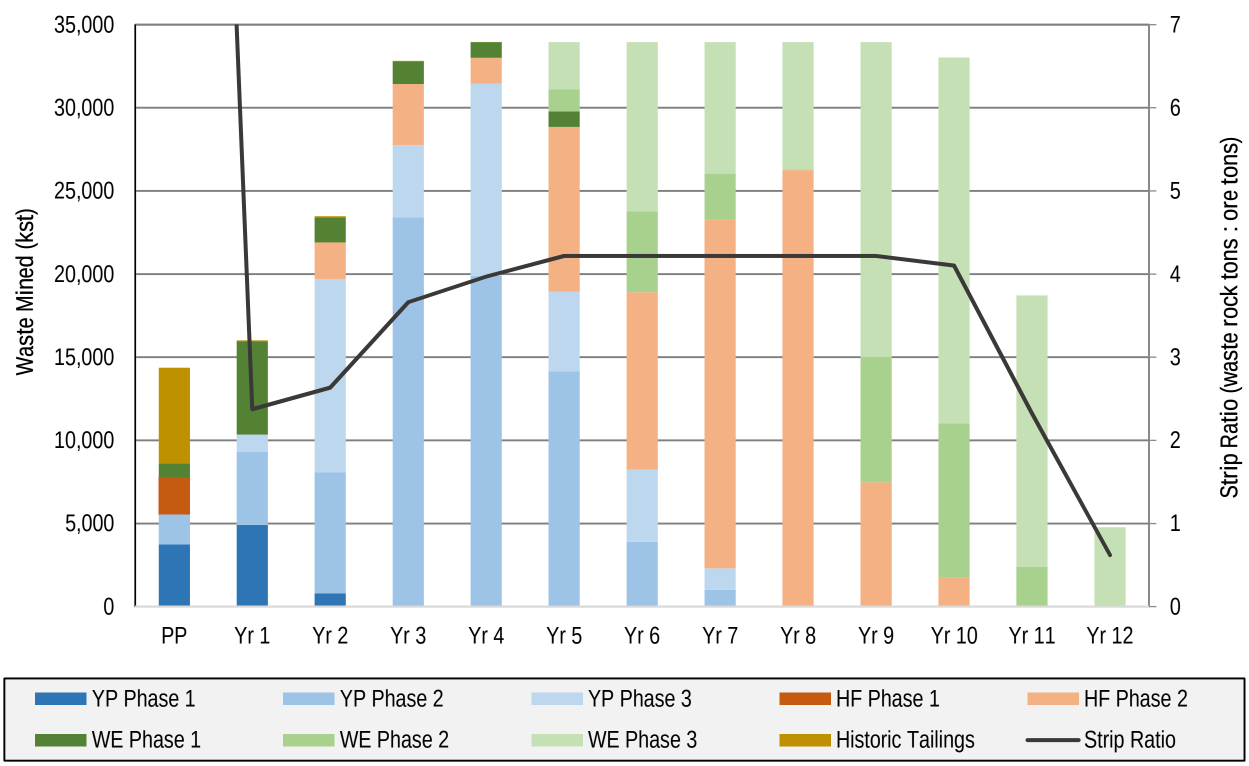


Figure 16.5: Waste Rock Mining Schedule by Deposit and Phase



16.4 MINE SCHEDULE

The mine schedule was developed based on the phase designs and the block models. The material contained within each pushback design was tabulated at multiple cutoff grades for input to the mine schedule process. As with the floating cone evaluation, only Indicated Mineral Resource categories were tabulated from the pushback designs. All other material (including Inferred Mineral Resources) was treated as waste rock in the mine schedule.

The mine schedule was developed to provide 8.05 million short tons of mined material to the primary crusher every year (22,050 short tons per day) after ramp up for approximately 12 years of mine life. The processing plant consists of a primary crusher, SAG mill, ball mill, two sequential flotation circuits, a POX circuit, CIP circuit, CIL circuit, and a conventional adsorption-desorption-recovery (ADR) plant. The designed process plant processes sulfide material to produce up to two mineral concentrate products: (1) when there is sufficient antimony grade to warrant, an antimony sulfide (stibnite) concentrate that would to be filtered, trucked off-site and sold; and, (2) for all material processed, an auriferous sulfide concentrate that would be oxidized onsite via POX, then processed via agitated leach, carbon stripping and refining to produce a gold- and silver-rich doré.

The Historic Tailings contain economic gold mineralization in the Indicated Mineral Resource category and are therefore included in the Mineral Reserve; they are planned to be sent to the grinding circuit during the first 4 years of the mine plan at a rate of 916 kst per year. The Historic Tailings would be mined via loader or excavator and trucked to a screening and re-slurrying system, then hydraulically transported to the process plant milling circuit. Since the Historic Tailings are quite fine-grained, typically 97% passing the #40-mesh, minimal incremental grinding effort is required by the process plant milling circuit; consequently, the processing plant would be able to accommodate the additional throughput from the Historic Tailings without the need to reduce the nominal daily mining rate.

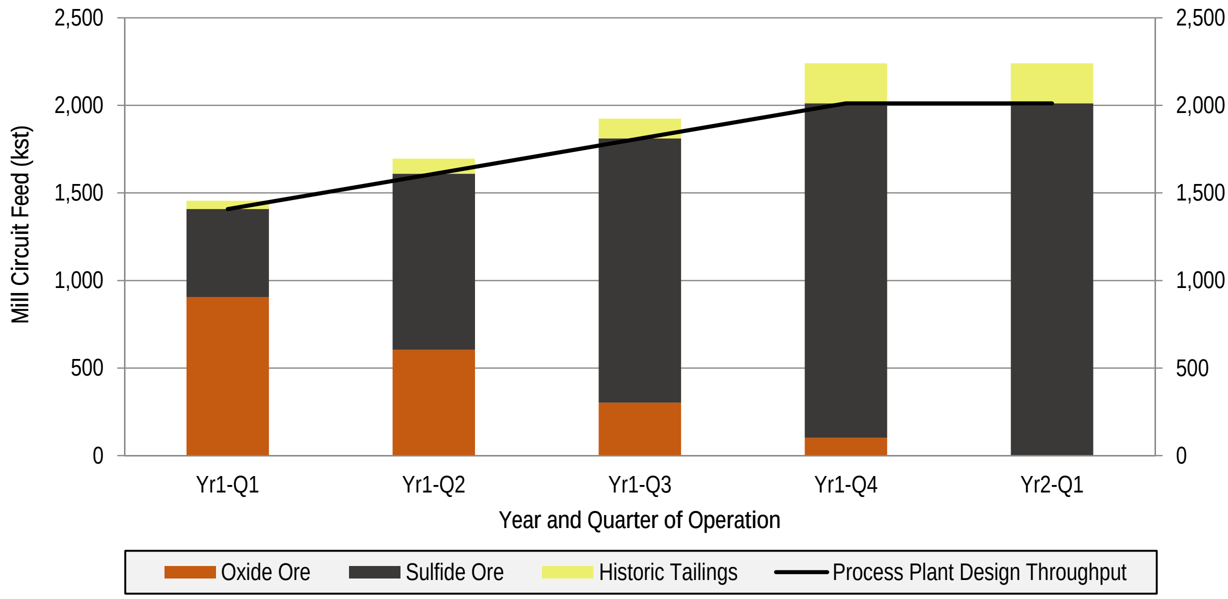
The process plant processes oxide material via crushing and grinding to produce a direct feed to the CIL circuit. Following plant commissioning, ramp up rates for the POX circuit are slower than the other circuits. To maximize utilization of all process circuits during the ramp-up period, oxide material from the West End open pit is stockpiled and batch-processed since it does not have to be processed through the POX circuit. The ramp-up schedules for the process plant and for the POX circuit are provided in Table 16.6 and Figure 16.6.

Table 16.6: Ramp-Up Schedule for Process Plant and Pressure Oxidation Circuit

Time Period	Overall Plant Design Throughput(1) (kst)	Process Plant Availability (%)	Process Plant Throughput(2) (kst)	POX Circuit Availability (%)	POX Circuit Throughput (kst)	Oxide Feed (kst)	Historic Tailings Feed (kst)
Yr1-Q1	2,012	70%	1,408	25%	503	905	48
Yr1-Q2	2,012	80%	1,609	50%	1,006	604	86
Yr1-Q3	2,012	90%	1,811	75%	1,509	302	113
Yr1-Q4	2,012	100%	2,012	95%	1,911	101	229
Yr2-Q1	2,012	100%	2,012	100%	2,012	-	229

Notes:
(1) Overall plant design throughput excludes Historical Tailings feed.
(2) Process plant availability and process plant throughput excludes the pressure oxidation circuit availability and throughput.

Figure 16.6: Chart of Ramp-Up Schedule for Process Plant and Pressure Oxidation Circuit



Following ramp-up, the oxidation circuit can be expected to have a lower annual average utilization when compared to the rest of the process plant. Fortunately, the West End deposit contains appreciable oxide mineralization that does not require oxidation prior to leaching; therefore, West End oxide material would be stockpiled and fed to the process plant during discrete times of the year while only the POX circuit is down for maintenance. This approach to processing the West End oxide material enables the process plant to operate at near its optimum utilization, while avoiding deferred processing of the high-grade, higher-NPR sulfide Mineral Reserves at Yellow Pine and Hangar Flats.

One of the main factors limiting the throughput of the POX circuit is the amount of oxygen that is able to diffuse into the slurry in the autoclave; the amount of sulfur in the feed concentrate determines how much oxygen is required to oxidize the ore. The sulfur levels are highest during the first two years of the mine schedule; however the recovery of sulfur to the antimony concentrate is expected to maintain the gold concentrate sulfur values at levels manageable for the POX circuit.

The high sulfur content of the mill feed in the first two years would produce lower than average pH solutions from the POX circuit; the low pH solutions would require additional neutralization material. Several options for neutralization material were evaluated and the selected option was to purchase additional ground lime in these periods to neutralize the tails. Other options evaluated included blending in West End ore, or adding crushed limestone from a marble deposit on the property. These options were cheaper, but they displaced enough high-grade mill feed to make them less economically attractive.

In the process of developing a sound mine operating strategy, multiple schedules were evaluated. The purpose of this work was to establish a cutoff grade schedule that balanced the increased revenue from higher head grades with the cost of mining additional waste rock tons. Using the design prices of $1,200/oz gold, $5/lb antimony and $23/oz silver, and a discount rate of 7%, a schedule was developed with increased cutoff grades and increased material mining rates to provide the highest NPV. While these design inputs differ from the final metal prices and discounting used in the PFS economic model, they were used to develop an understanding of the relative costs and benefits of increasing the head grades sent to the mill.

Cutoff grades were based on NPR in dollars per ton of ore ($/st ore) at the design prices stated above. NPR, defined as NSR less process plant OPEX and G&A, was calculated on a block-by-block basis in dollars per ton of ore ($/st ore) to indicate the value of a block.

NPR = NSR - Process Plant OPEX - Site G&A

Initially, mining costs were estimated to be $1.73/st of material. This would mean that the breakeven cutoff grade would be $1.73/st NPR and the internal cutoff grade would be $0.001/st NPR. The ore cutoff grade by period can be found in the second column of the Mine Production Schedule in Table 16.7.

Table 16.7 summarizes the mine production schedule that was developed for the PFS. This table represents the Mineral Reserve because the Probable Mineral Reserve corresponds to the total ore processed in the mine. The Probable Mineral Reserve category material was reported in Section 15. Figure 16.7 is a graphic summary of the material movements of ore and waste rock along with the strip ratio (waste rock tons : ore tons) by year and Figure 16.8 details the ore mined from each Mineral Reserve by year and includes the blended annual average gold head grade. Ore mined in pre-production (Year -1) is stockpiled and fed to the mill in Year 1.

Figure 16.7: Ore and Waste Rock Mined by Deposit by Year

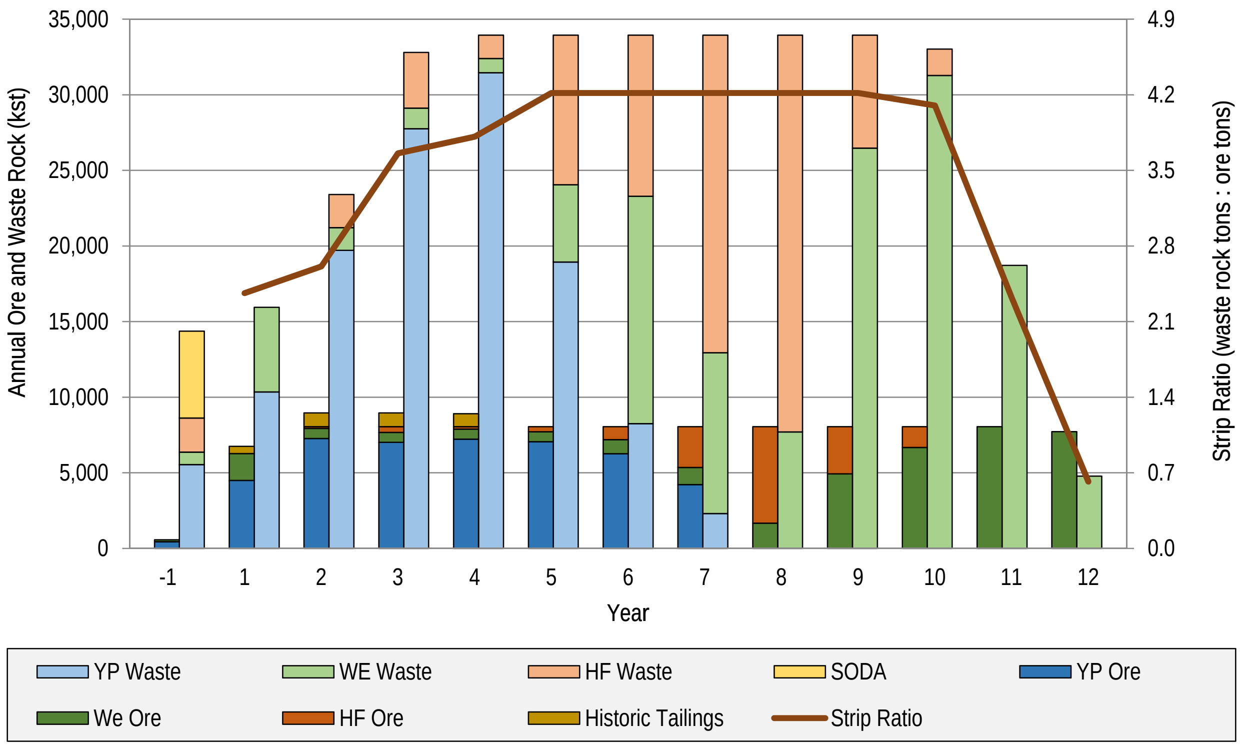


Figure 16.8: Ore Mined from Each Deposit by Type and by Year

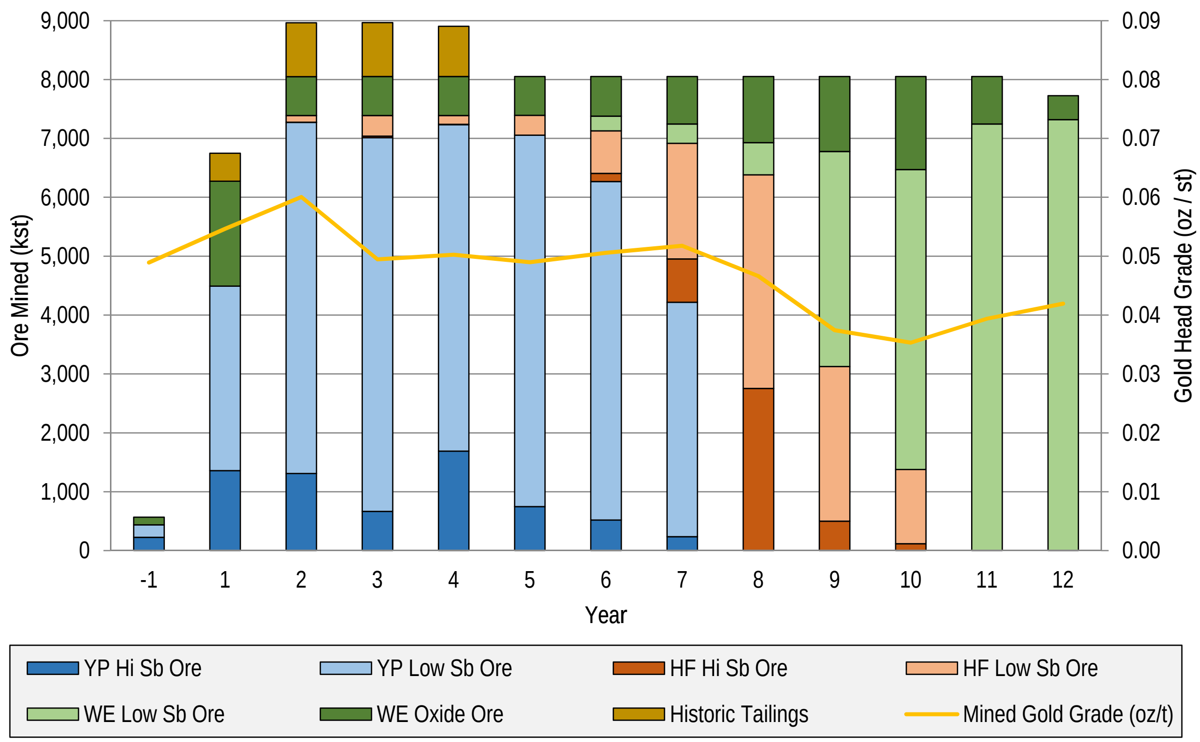


Table 16.7: Mine Production Schedule and Process Plant Feed Schedule

Time Period	Material Mined From All Pits								SODA	Ore Feed to Crusher					Historic Tailings Feed						Total Feed to the Process Plant				
	NPR Cutoff ($/st)	Ore (kst)	NPR ($/st)	Au (oz/st)	Ag (oz/st)	Sb (%)	Waste (kst)	Total (kst)	Waste (kst)	Ore (kst)	NPR ($/st)	Au (oz/st)	Ag (oz/st)	Sb (%)	Ore (kst)	NPR ($/st)	Au (oz/st)	Ag (oz/st)	Sb (%)	Waste (kst)	Ore (kst)	NPR ($/st)	Au (oz/st)	Ag (oz/st)	Sb (%)
Pre-prod-Q4							298	298	938																
Pre-prod-Q3	8.00	1	19.45	0.022	0.094	0.348	1,752	1,753	1,938																
Pre-prod-Q2	8.00	87	49.20	0.041	0.143	0.442	2,964	3,051	1,438																
Pre-prod-Q1	8.00	479	52.42	0.050	0.168	0.233	3,606	4,085	1,438																
Yr1-Q1	8.00	842	13.86	0.025	0.037	0.022	3,948	4,790		1,409	29.15	0.034	0.088	0.120	48	11.99	0.033	0.075	0.157	2	1,457	28.59	0.034	0.088	0.121
Yr1-Q2	8.00	1,609	50.01	0.050	0.119	0.183	4,348	5,957		1,609	50.01	0.050	0.119	0.183	86	11.85	0.033	0.075	0.159	6	1,695	48.07	0.049	0.116	0.182
Yr1-Q3	8.00	1,810	57.80	0.059	0.138	0.124	3,928	5,738		1,810	57.80	0.059	0.138	0.124	113	10.87	0.031	0.080	0.172	18	1,923	55.03	0.058	0.135	0.127
Yr1-Q4	8.00	2,012	65.44	0.066	0.133	0.109	3,725	5,737		2,012	65.44	0.066	0.133	0.109	229	16.33	0.037	0.080	0.163	35	2,241	60.42	0.063	0.128	0.115
Yr2-Q1	8.00	2,012	61.87	0.064	0.115	0.109	4,003	6,015		2,012	61.87	0.064	0.115	0.109	229	12.74	0.034	0.073	0.158	26	2,241	56.85	0.061	0.110	0.114
Yr2-Q2	8.00	2,013	55.24	0.058	0.099	0.108	4,980	6,993		2,013	55.24	0.058	0.099	0.108	229	2.13	0.023	0.048	0.092	25	2,242	49.82	0.054	0.094	0.107
Yr2-Q3	8.00	2,012	69.17	0.068	0.122	0.141	6,363	8,375		2,012	69.17	0.068	0.122	0.141	229	8.65	0.029	0.056	0.115	21	2,241	62.99	0.064	0.115	0.138
Yr2-Q4	7.00	2,012	46.06	0.052	0.070	0.060	8,064	10,076		2,012	46.06	0.052	0.070	0.060	229	19.22	0.040	0.089	0.178	1	2,241	43.32	0.051	0.072	0.072
Yr3	7.00	8,050	42.58	0.049	0.061	0.054	32,804	40,854		8,050	42.58	0.049	0.061	0.054	916	19.30	0.037	0.083	0.149	14	8,966	40.20	0.048	0.064	0.064
Yr4	2.00	8,050	48.51	0.050	0.108	0.155	33,950	42,000		8,050	48.51	0.050	0.108	0.155	692	15.69	0.032	0.112	0.226	15	8,742	45.91	0.049	0.108	0.160
Yr5	2.00	8,050	41.96	0.049	0.062	0.055	33,950	42,000		8,050	41.96	0.049	0.062	0.055							8,050	41.96	0.049	0.062	0.055
Yr6	3.00	8,050	44.19	0.051	0.059	0.058	33,950	42,000		8,050	44.19	0.051	0.059	0.058							8,050	44.19	0.051	0.059	0.058
Yr7	3.00	8,050	45.16	0.052	0.056	0.055	33,950	42,000		8,050	45.16	0.052	0.056	0.055							8,050	45.16	0.052	0.056	0.055
Yr8	3.00	8,050	42.52	0.046	0.100	0.171	33,950	42,000		8,050	42.52	0.046	0.100	0.171							8,050	42.52	0.046	0.100	0.171
Yr9	3.00	8,050	26.59	0.037	0.035	0.018	33,949	41,999		8,050	26.59	0.037	0.035	0.018							8,050	26.59	0.037	0.035	0.018
Yr10	4.00	8,050	24.31	0.035	0.048	0.006	33,021	41,071		8,050	24.31	0.035	0.048	0.006							8,050	24.31	0.035	0.048	0.006
Yr11	5.00	8,050	28.68	0.040	0.055	0.000	18,714	26,764		8,050	28.68	0.040	0.055	0.000							8,050	28.68	0.040	0.055	0.000
Yr12	2.00	7,726	30.18	0.042	0.045	0.000	4,778	12,504		7,726	30.18	0.042	0.045	0.000							7,726	30.18	0.042	0.045	0.000
Totals / Averages		95,065	40.31	0.047	0.071	0.067	340,995	436,060	5,752	95,065	40.31	0.047	0.071	0.067	3,001	14.96	0.034	0.084	0.165	163	98,066	39.53	0.047	0.071	0.070

Notes:
(1) NPR = Net of Process Revenue = Net Smelter Return ($/st ore) – Processing Costs ($/st ore) - Site G&A ($/st ore).
(2) Cutoff Grade for oxide ore from West End in years 1-8 is actually $0.001/st Net of Process.
(3) All units in table are imperial.
(4) Ore mined in pre-production is stockpiled and fed to the crusher in the first quarter of year 1.

16.5 WASTE ROCK STORAGE AND ALLOCATION

Waste rock from the three open pits is planned to be sent to four different destinations over the mine life. The four destinations include: the Main WRSF, the TSF embankment, the West End WRSF and the mined-out Yellow Pine open pit. The TSF embankment requires approximately 60,726 kst of rockfill for the bulk of its construction. The Main WRSF would then form a buttress immediately downstream of the TSF embankment. All of the Yellow Pine waste rock would be sent to the Main WRSF / TSF embankment, as would the Hangar Flats waste rock.

The Historic Tailings deposit lies within the footprint of the Main WRSF; consequently, the removal schedule of the Historic Tailings influences the construction of the Main WRSF. The Historic Tailings are planned to be removed from west to east; therefore, the WRSF would progress from the TSF embankment towards the east as the tailings are removed from the toe of the WRSF. West End waste rock would be sent to the West End WRSF for the first 6 years of mine life until the mining of the Yellow Pine open pit is complete; after Year 6 West End waste rock is used to backfill the Yellow Pine open pit. Figure 16.9 and Table 16.8 summarize the origin and destination of mine waste rock by time period.

Figure 16.9: Waste Rock Origin by Deposit by Year

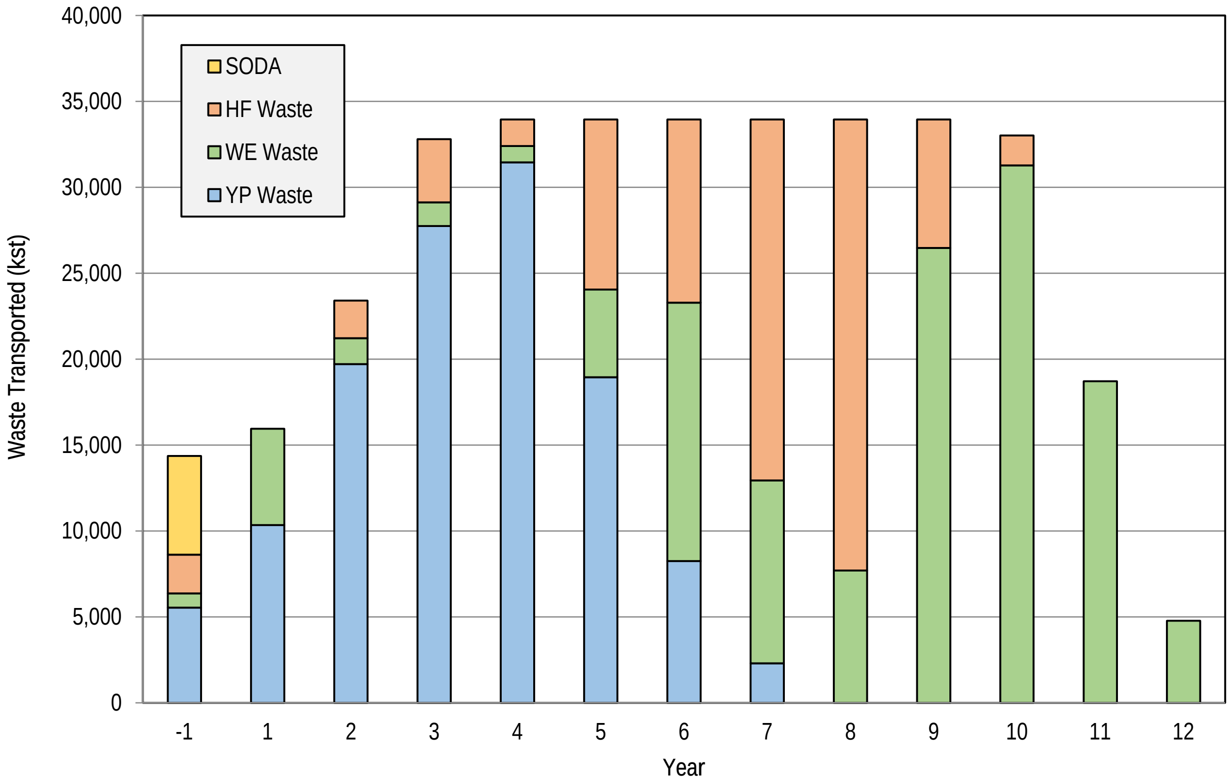


Figure 16.10: Waste Rock Destination by Period

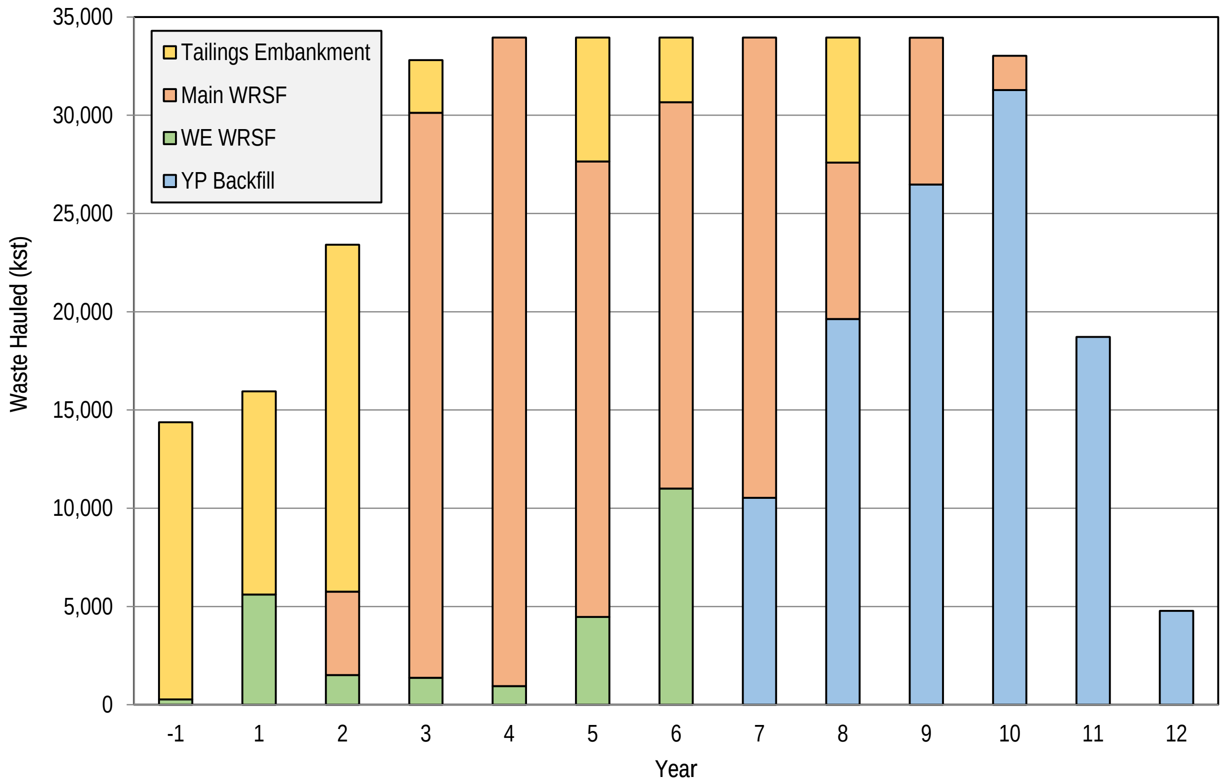


Table 16.8: Waste Rock Destination by Period

Time Period	Tailings Embankment (kst)	Main WRSF (kst)	West End WRSF (kst)	Yellow Pine Backfill (kst)
Pre-prod-q-4	1,236	-	-	-
Pre-prod-q-3	3,690	-	-	-
Pre-prod-q-2	4,402	-	-	-
Pre-prod-q-1	4,769	-	275	-
yr1-q1	1,420	-	2,528	-
yr1-q2	2,367	-	1,981	-
yr1-q3	3,042	-	886	-
yr1-q4	3,514	-	211	-
yr2-q1	4,003	-	-	-
yr2-q2	4,288	-	692	-
yr2-q3	6,363	-	-	-
yr2-q4	3,000	4,248	816	-

Time Period	Tailings Embankment (kst)	Main WRSF (kst)	West End WRSF (kst)	Yellow Pine Backfill (kst)
yr3	2,683	28,747	1,374	-
yr4	-	33,004	946	-
yr5	6,300	23,185	4,465	-
yr6	3,289	19,661	11,000	-
yr7	-	23,420	-	10,530
yr8	6,360	7,960	-	19,630
yr9	-	7,479	-	26,470
yr10	-	1,744	-	31,277
yr11	-	-	-	18,714
yr12	-	-	-	4,778
Total	60,726	149,448	25,174	111,399

As discussed earlier, the Historic Tailings are overlain by neutralized SODA. The spent ore material is planned to be loaded and hauled to the TSF embankment during pre-production as the physical characteristics and location of the material are well suited for TSF embankment construction. Approximately 5,752 kst of SODA material would be moved during the pre-production period.

16.6 MINE OPERATIONS AND EQUIPMENT

Mine mobile equipment was selected to meet the production requirements summarized in Table 16.7. All mine equipment selected for this study is standard off-the-shelf units, with the exception of the haul truck beds. Lightweight truck beds were chosen to enable the trucks to load and haul more material. Although the lightweight beds need to be replaced more frequently, the additional cost of the beds is compensated by the increased truck productivity.

Mining is scheduled for 365 days/year and 2 shifts/day of 12 hours duration. Twenty shifts per year are assumed to be lost due to weather delays and holidays. A 4-crew system working 14 days on and 14 days off has been used when calculating mine equipment operators and maintenance personnel.

The majority of production drilling is planned to be accomplished with conventional track mounted rotary blast-hole drills. Drills were selected based on the physical characteristics of the Mineral Resource and the required mining rate, and would have a 60,0000-lb pull down force with a 7-7/8" bit diameter. All dry holes would be loaded with ammonium nitrate-fuel oil (ANFO) while wetter holes would be lined with a plastic liner before they are loaded with emulsion slurry.

The majority of production loading is planned with 23.5 cubic yard frontend loaders. Wheel loaders were chosen over shovels for economic reasons and because the maneuverability would be beneficial at the Stibnite Gold Project since mining occurs in three separate pits. For several years, all three pits are being mined simultaneously. The 23.5 cubic yard loaders are also well suited for snow clearing and loading, SODA material loading, and to feed the crusher from stockpiles when trucked ore cannot meet the process plant throughput requirements.

Hauling is planned to be accomplished with 200-ton haul trucks fitted with lightweight beds (carrying 219 tons) for a majority of the ore and waste rock. The 200-ton haul trucks would also be used to haul accumulated snow out of the open pits and to haul SODA material to the tailings embankment.

Other equipment selected for the mining fleet include: 580-hp (D10 class) track dozers; graders with 16-foot moldboards, 20,106-gallon water trucks on 100-ton haul truck chassis, and a Low Ground Pressure 200-hp dozer to support the removal of the Historic Tailings. Small track mounted drills are included in the equipment requirements for secondary blasting and road pioneering duties. Also, these small drills would do some production drilling on the very highest benches of the phases where the working areas would not be large enough for the main production fleet. Two 4-yard backhoes would be used for general support and maintenance of drainage structures and are also planned to be used for loading Historic Tailings during the first four years of mine life.

A small fleet of 40-ton articulated haul trucks are planned for the Project; these trucks would be used for hauling Historic Tailings, constructing haul roads, and hauling ore and waste rock from the highest benches of the mine phases when the working room is narrow. A smaller 10.5-yard loader is also planned for loading the 40-ton haul trucks on the high benches and for assisting in haul road construction.

Equipment productivity was calculated on a per-shift basis considering the Project material and operating conditions. The productivity per shift and the tonnage requirements set the number of operating shifts needed per year to move the required material. Availability and utilization were applied to determine the required number of operating units. Haul truck productivity was based on detailed haul time simulations over measured haul profiles. Haul profiles were measured for each material type by time period, from each phase and storage location to each destination. Table 16.9 summarizes the mine mobile equipment fleet requirements for the mine life. In some years the mobile equipment on hand may be greater than the average fleet required; this results from the need to account for short-term fluctuations in equipment requirements.

Table 16.9: Major Mine Mobile Equipment Requirements

Equipment Type	Time Period												
	PPQ -5	PPQ -4	PPQ -3	PPQ -2	PPQ -1	Yr1-Q1	Yr1-Q2	Yr1-Q3	Yr1-Q4	Yr2-Q1	Yr2-Q2	Yr2-Q3	Yr2-Q4
Cat MD6290 Blasthole Drill	0	0	0	1	1	2	3	3	3	3	3	4	4
Cat 994 Loader	0	1	2	2	2	2	3	3	2	2	3	4	4
Cat 789 Haul Truck	0	3	4	6	10	6	9	10	10	11	14	17	18
Cat D10 Track Dozer	3	5	5	4	5	4	4	4	4	4	5	4	4
Cat D6TLGP Dozer	0	0	0	0	0	1	1	1	1	1	1	1	1
Cat 16M Grader	2	2	2	2	2	3	3	3	3	3	3	3	3
Cat 777 Water Truck	0	1	1	1	2	2	2	2	2	2	2	2	2
Cat 990 Loader	1	1	1	1	1	1	1	1	1	1	1	1	1
Cat 740 Haul Truck	3	5	5	5	3	3	5	4	5	5	4	3	3
Cat MD 5150 Pioneer Drill	3	3	3	2	2	2	2	2	2	1	2	1	1
Cat 349 Excavator	1	1	1	1	1	1	2	2	2	2	2	2	2
TOTAL	13	22	24	25	29	27	35	35	35	35	40	42	43

Equipment Type	Time Period									
	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
Cat MD6290 Blasthole Drill	5	5	5	5	5	5	5	5	3	2
Cat 994 Loader	4	4	4	4	4	4	4	4	3	2
Cat 789 Haul Truck	18	20	19	20	15	16	16	13	9	6
Cat D10 Track Dozer	4	5	4	3	3	3	3	3	2	2
Cat D6TLGP Dozer	1	1	0	0	0	0	0	0	0	0
Cat 16M Grader	3	3	3	3	3	3	3	2	2	1
Cat 777 Water Truck	2	2	2	2	2	2	2	2	1	1
Cat 990 Loader	1	1	1	1	1	1	1	1	1	1
Cat 740 Haul Truck	3	3	3	2	1	1	5	5	1	1
Cat MD 5150 Pioneer Drill	1	1	2	1	1	1	1	1	1	1
Cat 349 Excavator	2	2	1	1	1	1	1	1	1	1
TOTAL	44	47	44	42	36	37	41	37	24	18

The requirements for mine supervision, operations, and maintenance personnel were calculated using the equipment list and mine schedule. For the first half of the mine life 36 salaried personnel were included for supervision, engineering, geology, and ore control; starting in Year 7, only 34 salaried personnel were included.

Mine operations and maintenance labor increases to 213 persons in the end of year two and stays between 213 and 230 persons until labor requirements begin to decline in Year 7. Maintenance personnel requirements are set to be around 50% of operations labor required. The salary and hourly staff requirements are provided in Table 16.10 and Table 16.11, respectively. Figure 16.11 presents the mine staffing graphically.

Table 16.10: Salary Staff Requirements

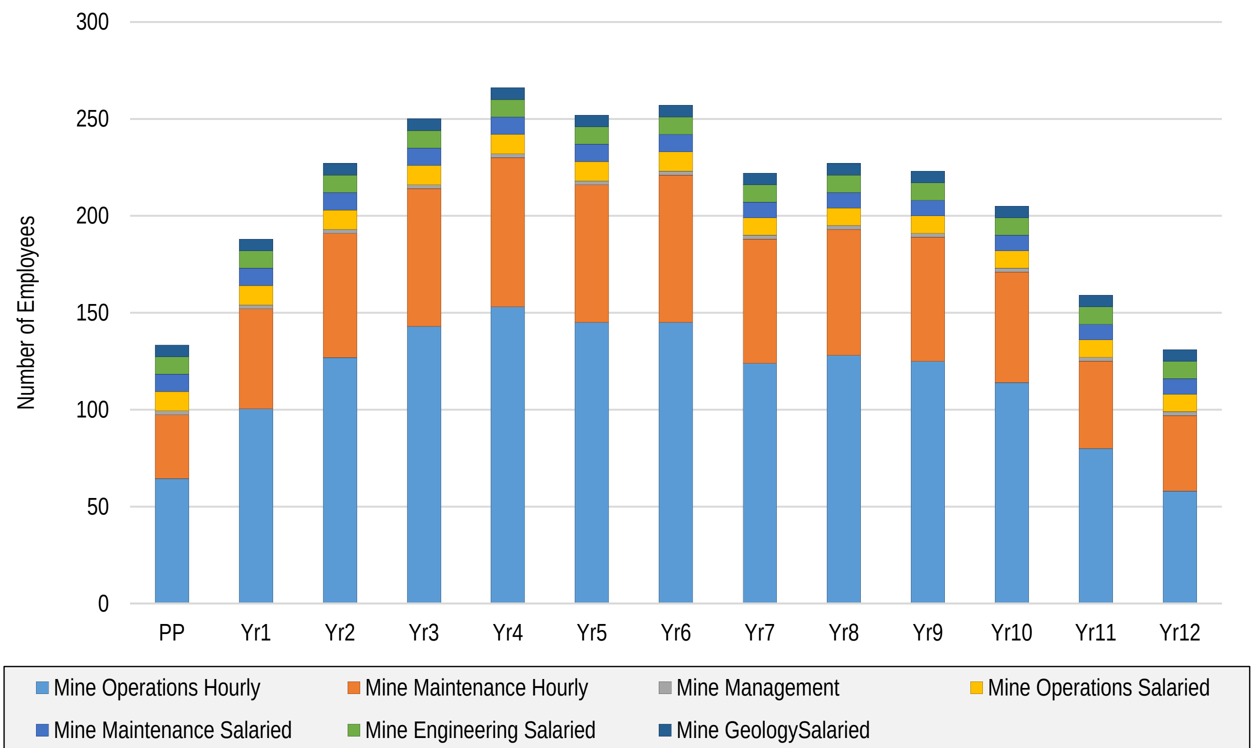
Job Titles	Time Period												
	Pre-Prod	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1	1
Secretary	1	1	1	1	1	1	1	1	1	1	1	1	1
Total	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Operations													
Mine Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Shift Foreman	4	4	4	4	4	4	4	4	4	4	4	4	4
Blasting Foreman	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Clerk	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Trainer	1	1	1	1	1	1	1	-	-	-	-	-	-
Mine Operations Total	10	10	10	10	10	10	10	9	9	9	9	9	9
Mine Maintenance													
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Shift Foreman	4	4	4	4	4	4	4	4	4	4	4	4	4
Maintenance Planner	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Trainer	1	1	1	1	1	1	1	-	-	-	-	-	-
Maintenance Clerk	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Maintenance Total	9	9	9	9	9	9	9	8	8	8	8	8	8
Mine Engineering													
Senior Mine Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1
Mining Engineer	2	2	2	2	2	2	2	2	2	2	2	2	2
Surveyor	2	2	2	2	2	2	2	2	2	2	2	2	2
Surveyor Helper	4	4	4	4	4	4	4	4	4	4	4	4	4
Mine Engineering Total	9	9	9	9	9	9	9	9	9	9	9	9	9
Mine Geology													
Senior Mine Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Geologist	2	2	2	2	2	2	2	2	2	2	2	2	2
Geotechnical Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1
Sampler	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Geology Total	6	6	6	6	6	6	6	6	6	6	6	6	6
Total Personnel	36	36	36	36	36	36	36	34	34	34	34	34	34

Table 16.11: Hourly Staff Requirements

Job Titles	Time Period												
	PPQ -5	PPQ -4	PPQ -3	PPQ -2	PPQ -1	Yr1-Q1	Yr1-Q2	Yr1-Q3	Yr1-Q4	Yr2-Q1	Yr2-Q2	Yr2-Q3	Yr2-Q4
Mine Operations													
Drill Operator	-	-	-	1	3	6	8	7	7	7	8	10	13
Loader Operator	-	2	5	5	6	5	7	7	6	6	9	10	10
Haul Truck Driver	-	4	13	19	31	17	27	33	33	35	43	54	56
Track Dozer Operator	4	6	12	11	11	11	10	10	10	10	11	10	10
LGP Dozer Operator	-	-	-	-	-	1	1	1	2	2	2	2	2
Grader Operator	2	3	4	6	5	6	6	6	7	7	7	8	7
Service Crew	10	15	26	26	26	26	26	26	26	26	26	26	26
Blasting Crew	4	6	6	6	6	6	6	6	6	6	6	6	6
Dispatch Operator	-	-	4	4	4	4	4	4	4	4	4	4	4
Laborer	2	6	6	6	6	6	6	6	6	6	6	6	6
Mine Operations Totals	22	42	76	84	98	88	101	106	107	109	122	136	140
Mine Maintenance													
Mechanic	2	6	11	13	17	14	19	20	20	20	24	29	31
Mechanic's Helper	1	3	4	5	7	6	8	8	8	8	9	11	12
Welder	1	2	4	4	5	5	6	6	6	6	8	9	10
Fuel & Lube Crew	4	4	8	8	8	8	8	8	8	8	8	8	8
Tire Crew	4	4	8	8	8	8	8	8	8	8	8	8	8
Laborer	2	2	4	4	4	4	4	4	4	4	4	4	4
Mine Maintenance Totals	14	21	39	42	49	45	53	54	54	54	61	69	73
Total Labor Requirements	36	63	115	126	147	133	154	160	161	163	183	205	213
Maintenance / Operations Ratio	0.64	0.50	0.51	0.50	0.50	0.51	0.52	0.51	0.50	0.50	0.50	0.51	0.52

Job Titles	Time Period									
	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
Mine Operations										
Drill Operator	14	15	15	15	13	13	14	14	10	5
Loader Operator	11	11	11	11	11	11	10	10	7	4
Haul Truck Driver	57	64	60	63	46	50	50	42	28	17
Track Dozer Operator	10	11	10	7	6	6	6	6	5	5
LGP Dozer Operator	2	2	0	0	0	0	0	0	0	0
Grader Operator	7	8	7	7	6	6	6	5	5	2
Service Crew	26	26	26	26	26	26	23	21	9	9
Blasting Crew	6	6	6	6	6	6	6	6	6	6
Dispatch Operator	4	4	4	4	4	4	4	4	4	4
Laborer	6	6	6	6	6	6	6	6	6	6
Mine Operations Totals	143	153	145	145	124	128	125	114	80	58
Mine Maintenance										
Mechanic	30	34	30	33	26	27	26	22	14	11
Mechanic's Helper	12	13	12	13	10	10	10	8	6	4
Welder	9	10	9	10	8	8	8	7	5	4
Fuel & Lube Crew	8	8	8	8	8	8	8	8	8	8
Tire Crew	8	8	8	8	8	8	8	8	8	8
Laborer	4	4	4	4	4	4	4	4	4	4
Mine Maintenance Totals	71	77	71	76	64	65	64	57	45	39
Total Labor Requirements	214	230	216	221	188	193	189	171	125	97
Mine Maintenance/Operations Ratio	0.50	0.50	0.49	0.52	0.52	0.51	0.51	0.50	0.56	0.67

Figure 16.11: Salaried and Hourly Mining Personnel by Department by Year



16.7 EXTERNAL HAUL ROADS AND MINE SEQUENCE DRAWINGS

The terrain in the area of the Stibnite Gold Project comprises steep-walled valleys and, as a result, initial haul road access to the upper benches of the open pits would require significant effort in road pioneering. Construction of these roads is planned ahead of phase mining so that access is available during scheduled mining. Designs of the initial access roads and other necessary external haul roads can be seen on the time sequence plans presented on Figure 16.12 to Figure 16.24, inclusively. Key details for each year of mining are provided with each figure.

Mining at the Project would begin in the Yellow Pine Deposit to target the lowest cost gold ounces. Yellow Pine is scheduled to be mined as quickly as possible because it contains the lowest cost ounces and also because the Yellow Pine pit needs to be available for backfilling with waste rock generated from West End, and to ultimately re-route the EFSFSR to its pre-mining vertical and horizontal alignments. The Yellow Pine pit is completed midway through year 7, at which time it begins to be backfilled with West End waste rock.

The mill requires 1,912,000 tons of oxide in the first year of production for plant ramp up. Following year 1, the mill requires at least 660,000 tons of oxide ore per year for processing during scheduled autoclave maintenance periods. A small initial phase targeting oxides is designed in the West End deposit, which contains the only oxide resource of the Project. This small phase contains enough oxide ore to feed the mill for five years before oxide ore is released from later West End phases. Waste produced from West End before the Yellow Pine pit is available for backfilling is stored in a small waste rock storage facility up the canyon from the West End open pit.

During pre-production, 8,710 kst of rock fill are required for the construction of the TSF starter embankment. This rock requirement necessitates mining more than just the primary Yellow Pine phase during pre-production. Waste rock is mined from Yellow Pine phases and a small phase in Hangar Flats to combine with SODA material to make up the construction requirements of the TSF in pre-production.

The Main WRSF is located east and on the down-slope of the TSF embankment. It expands eastward from the TSF as the Historic Tailings are removed and reprocessed, ensuring additional buttressing of the TSF. Once the Historic Tailings are completely removed in year four, the Main WRSF is expanded to the final footprint so that the waste rock can be placed in lifts from lower to higher elevations. During mining, Yellow pine waste rock is planned to be preferentially sent to the lower elevation of the WRSF over the TSF except when the TSF embankment requires additional construction material.

Figure 16.12: Annual Open Pit and Waste Rock Storage Facility Plan – End of Pre-Production

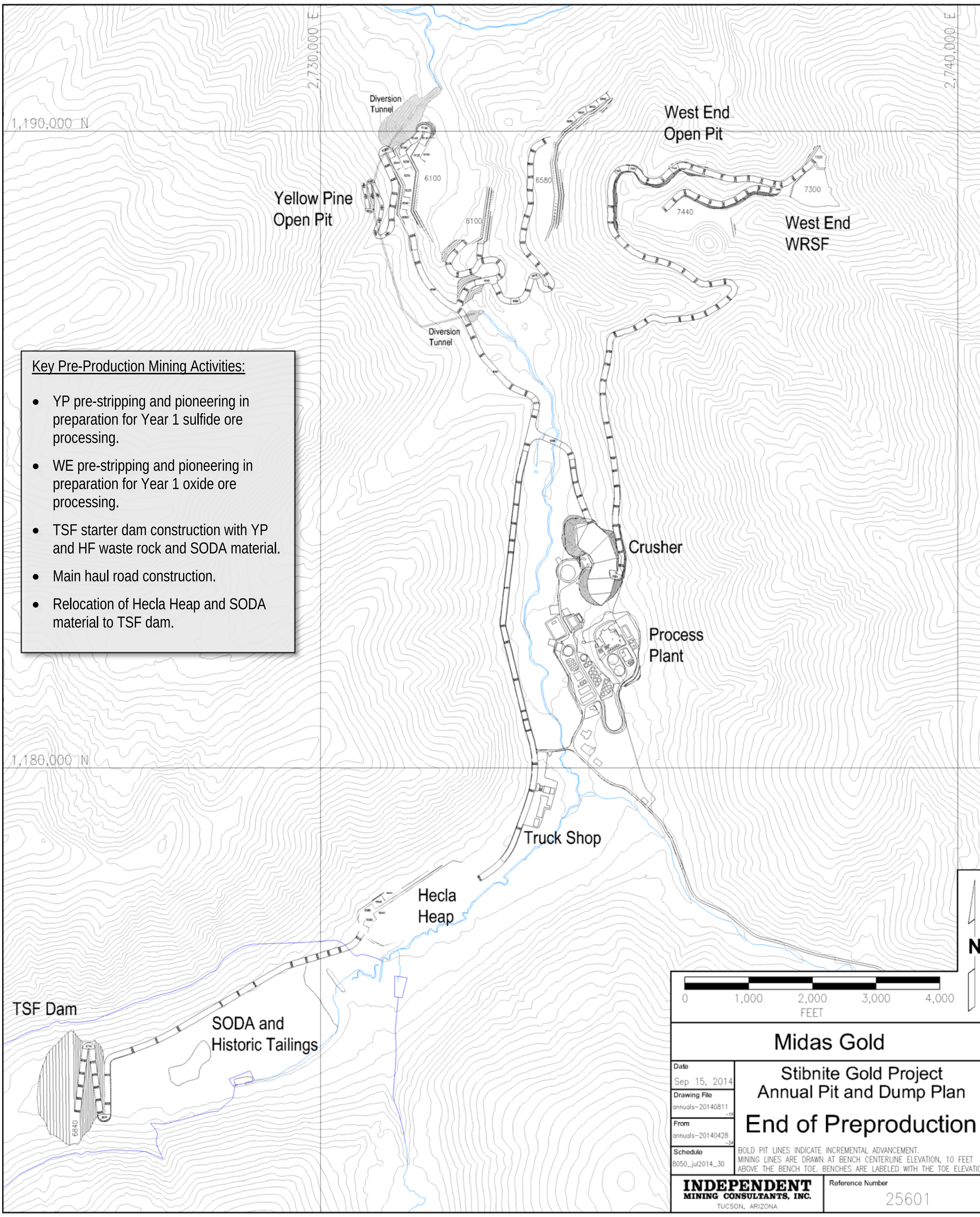


Figure 16.13: Annual Open Pit and Waste Rock Storage Facility Plan – End of Year 1

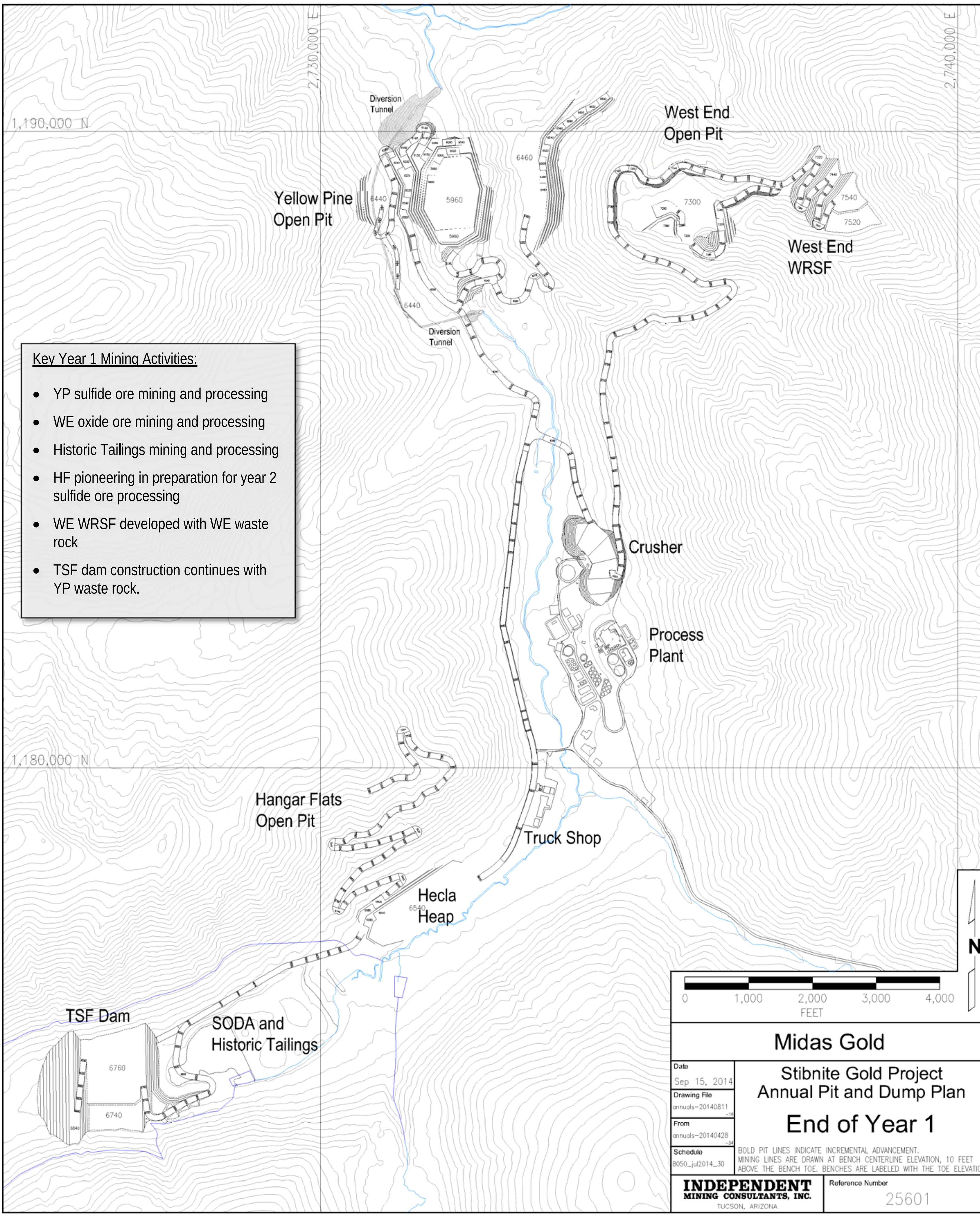


Figure 16.14: Annual Open Pit and Waste Rock Storage Facility Plan – End of Year 2

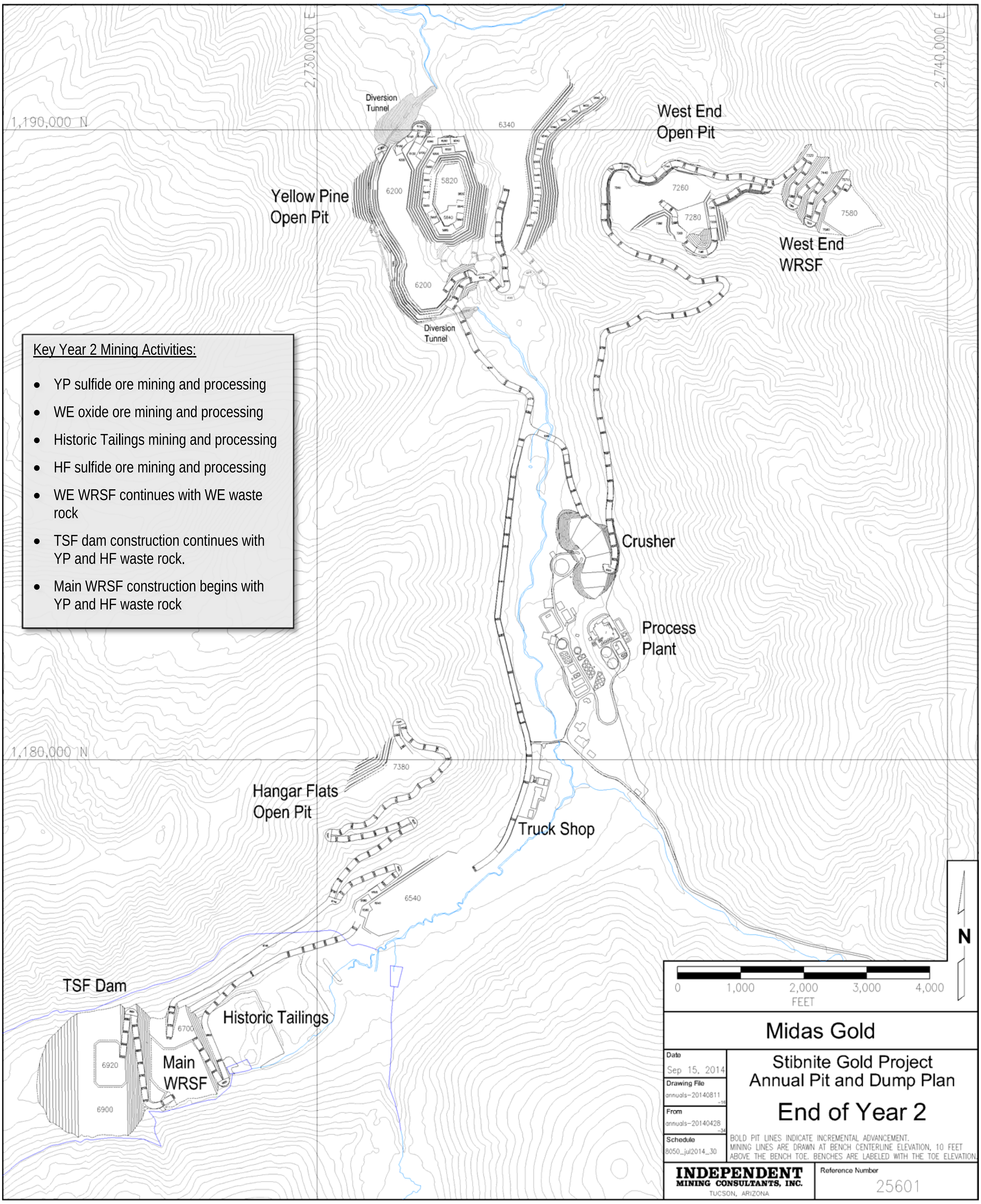


Figure 16.15: Annual Open Pit and Waste Rock Storage Facility Plan – End of Year 3

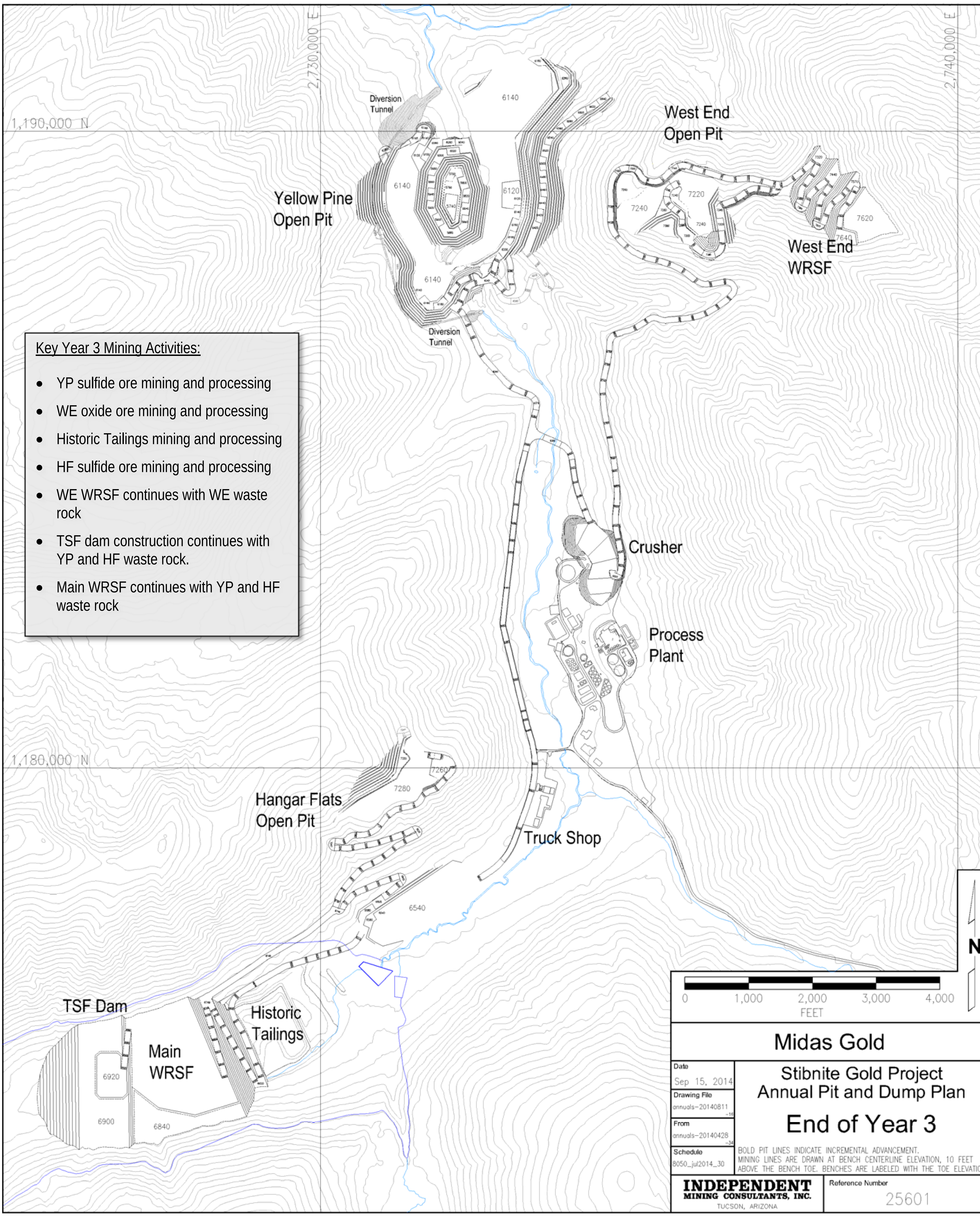


Figure 16.16: Annual Open Pit and Waste Rock Storage Facility Plan – End of Year 4

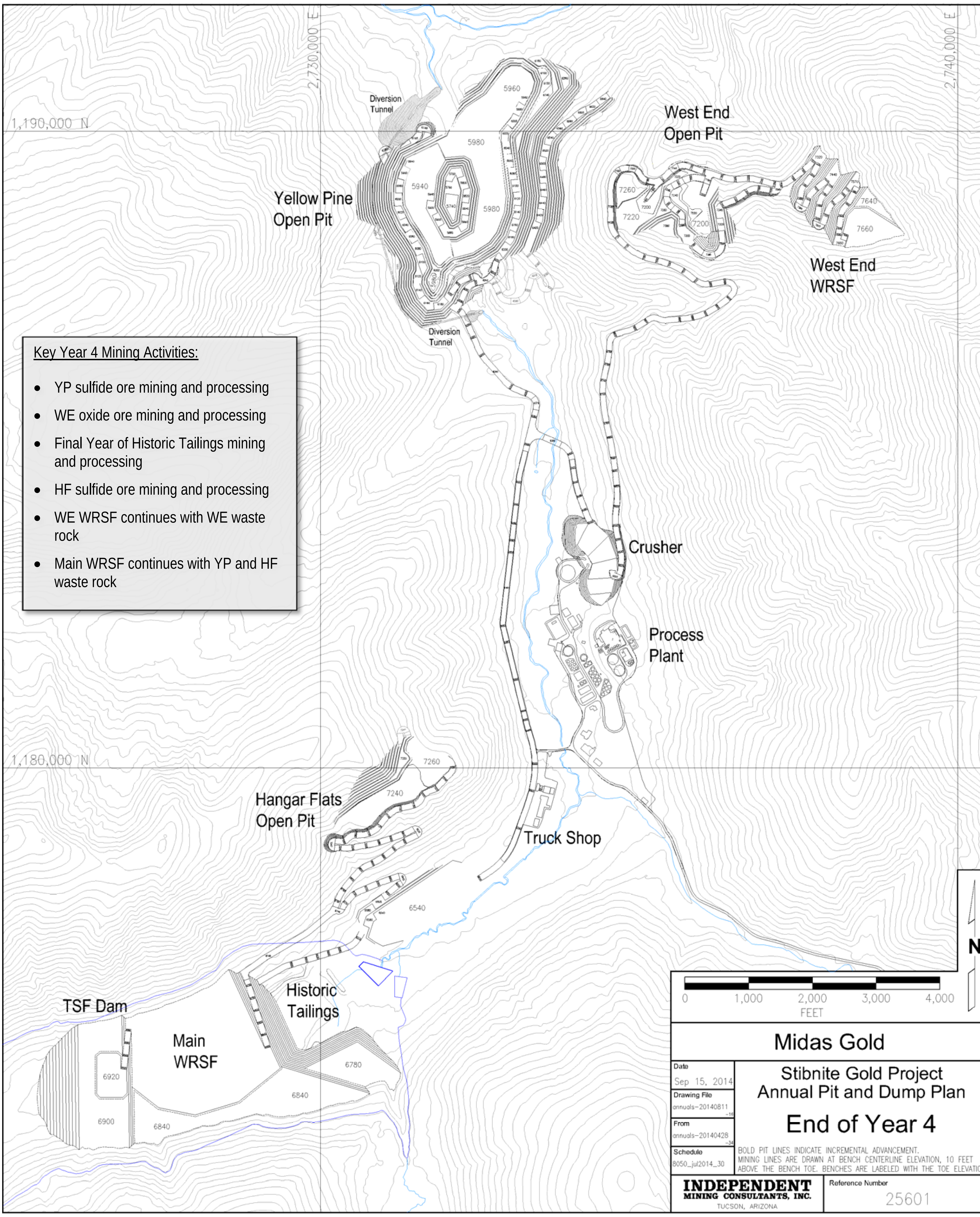


Figure 16.17: Annual Open Pit and Waste Rock Storage Facility Plan – End of Year 5

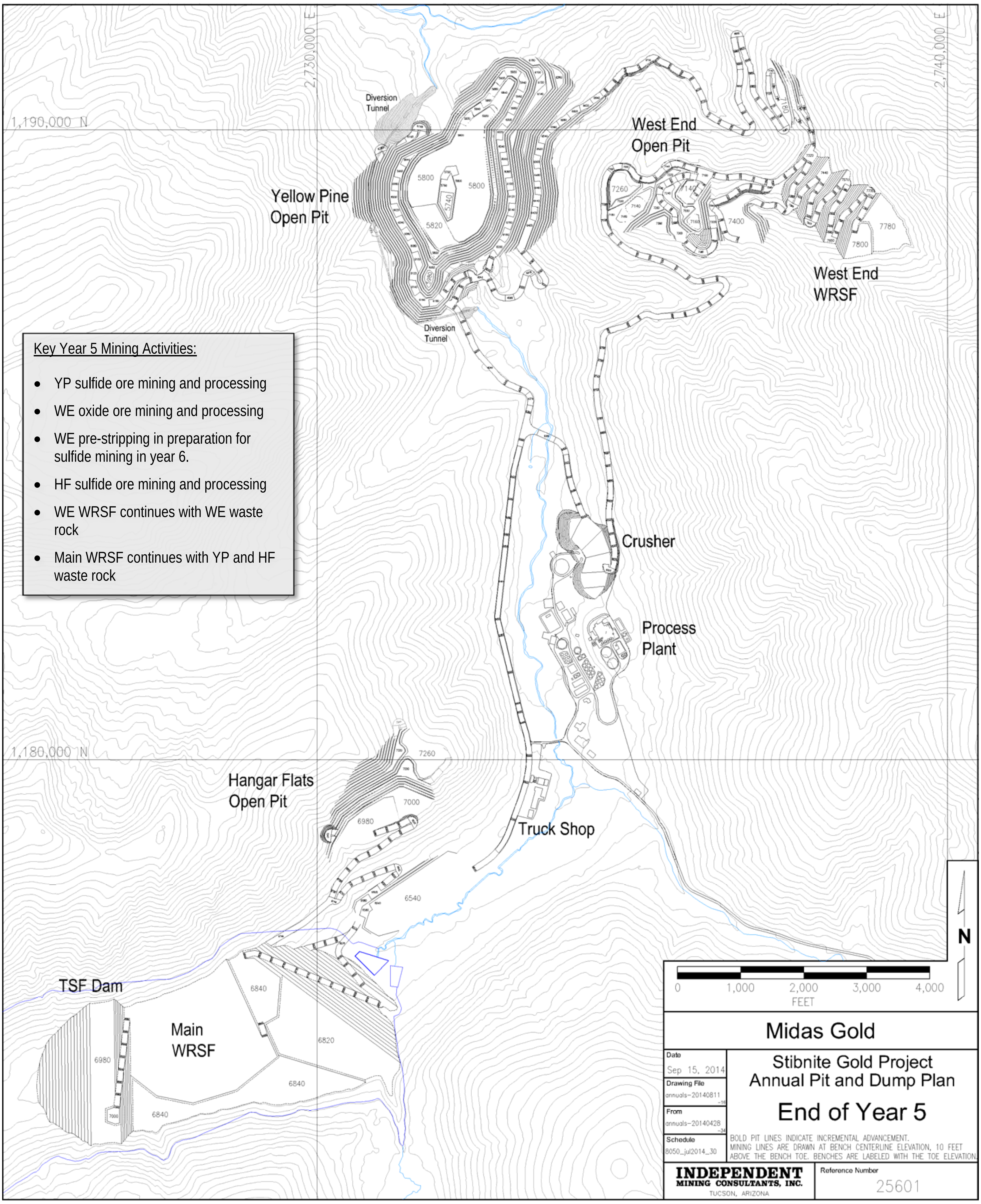


Figure 16.18: Annual Open Pit and Waste Rock Storage Facility Plan – End of Year 6

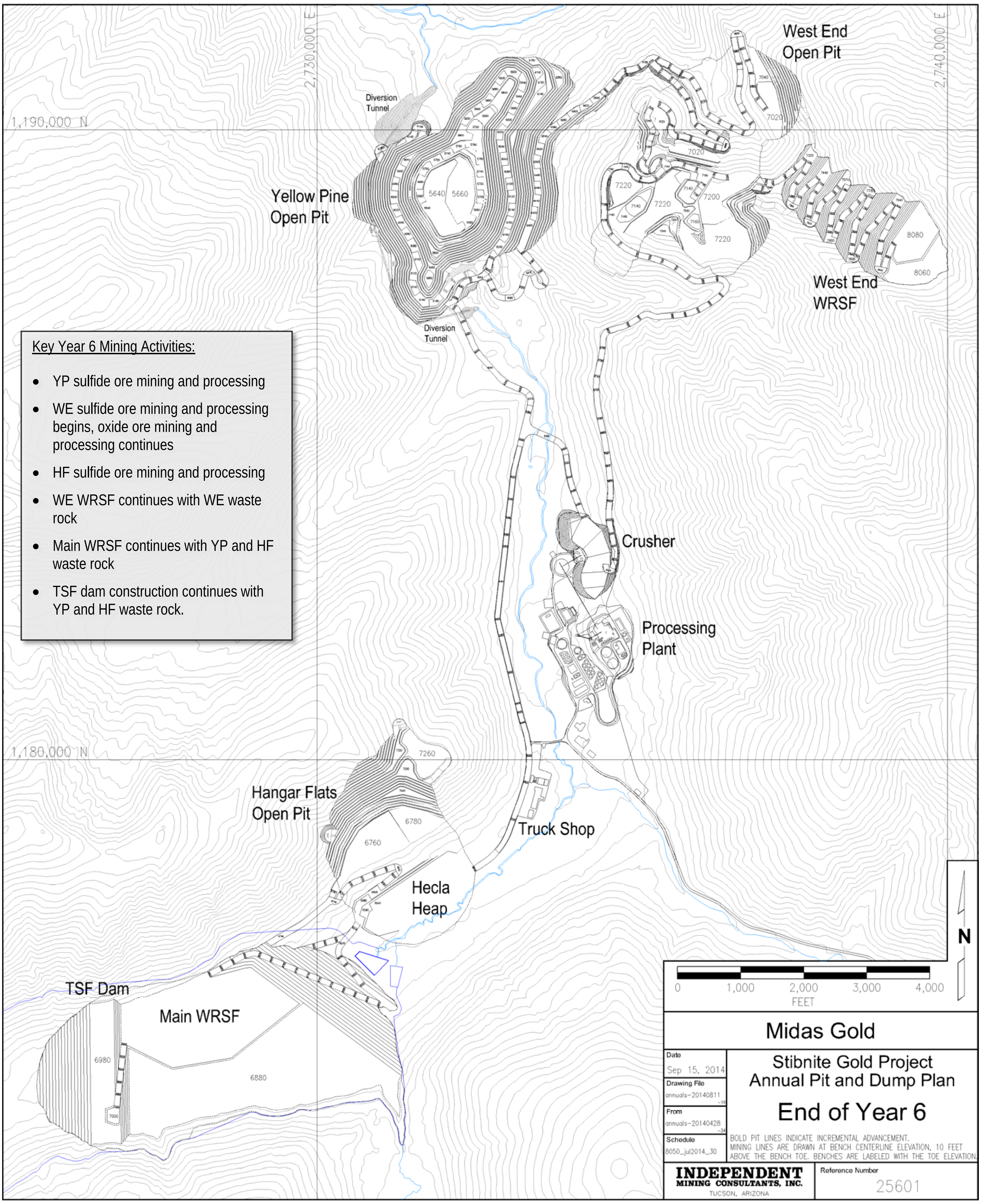


Figure 16.19: Annual Open Pit and Waste Rock Storage Facility Plan – End of Year 7

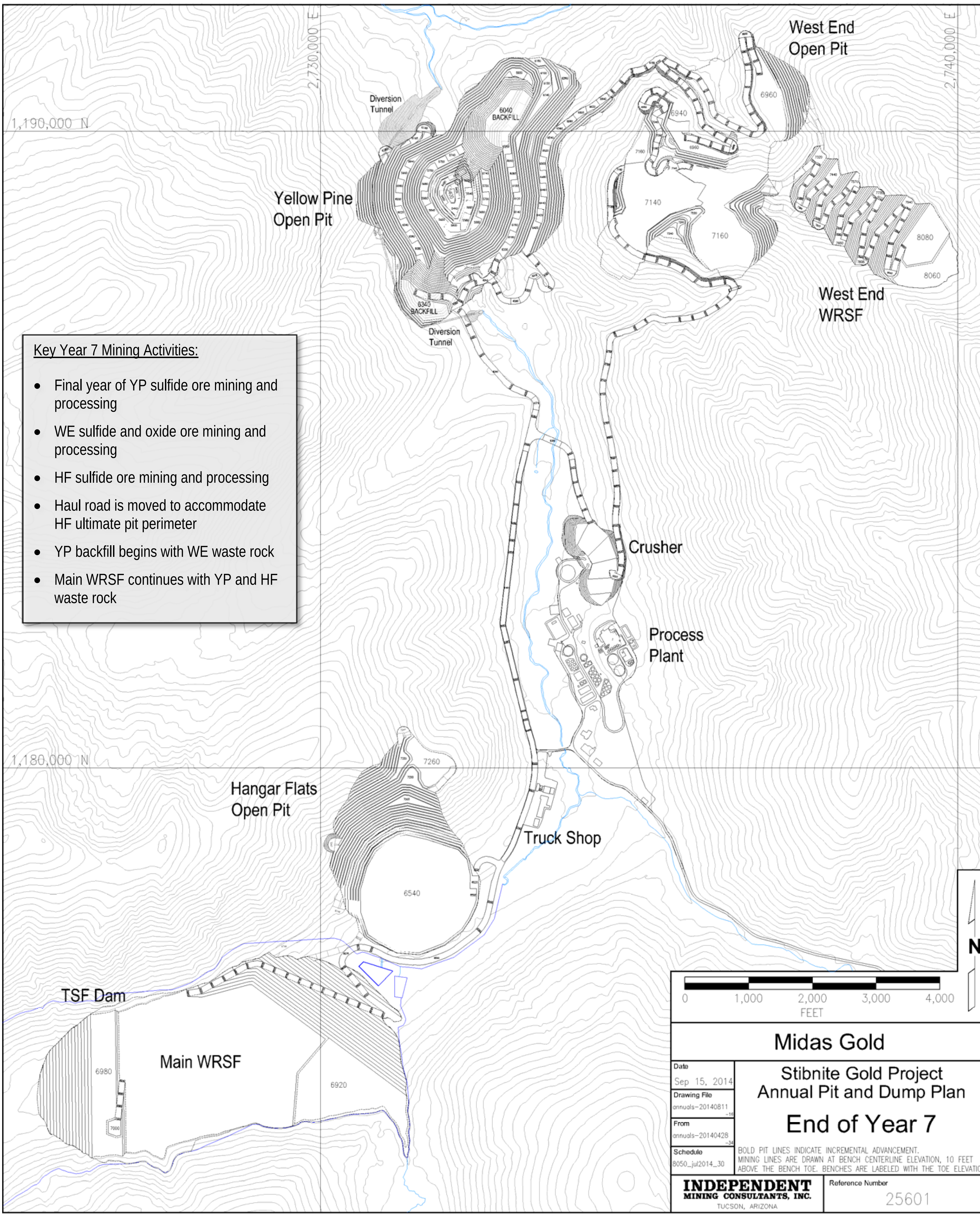


Figure 16.20: Annual Open Pit and Waste Rock Storage Facility Plan – End of Year 8

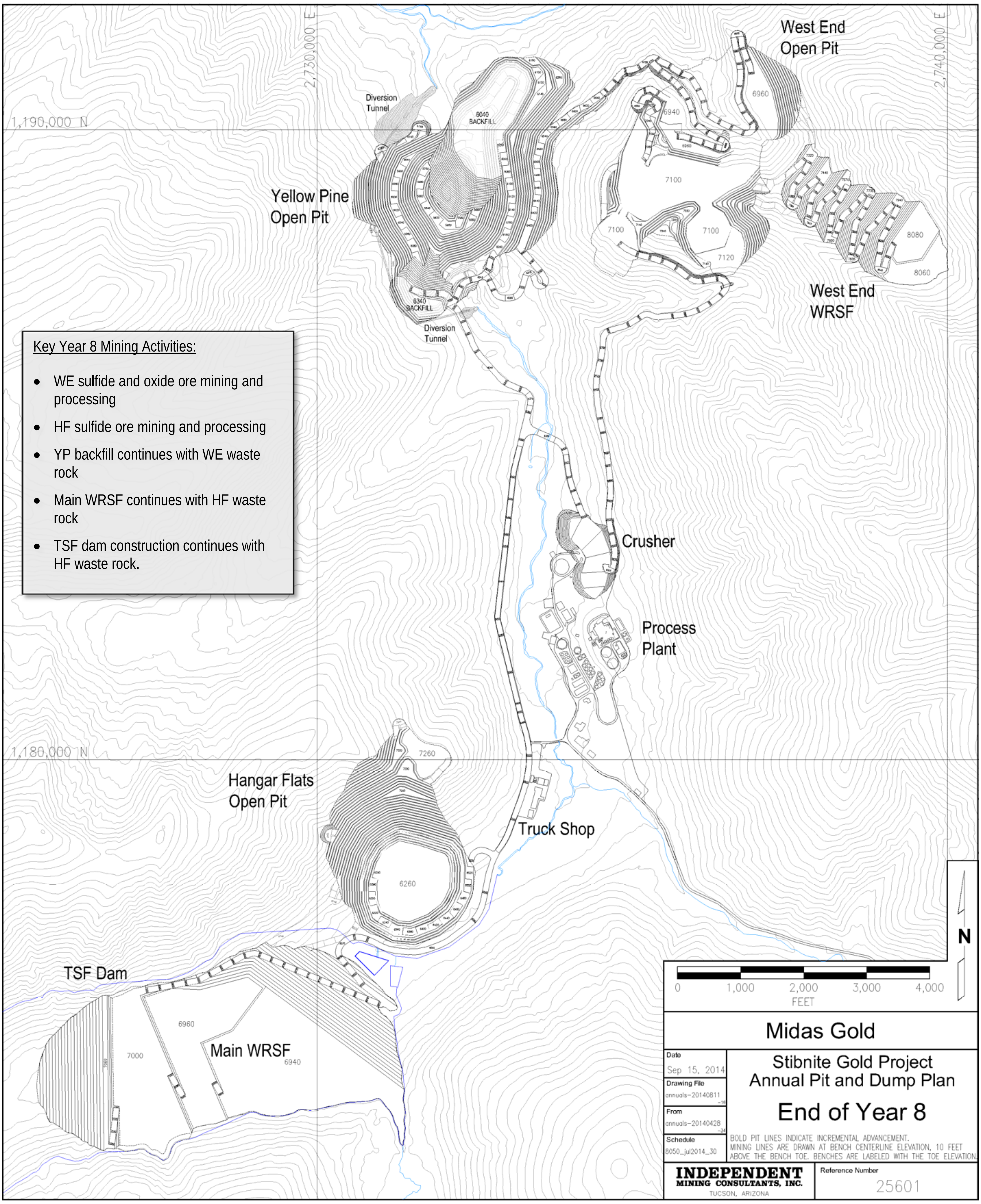


Figure 16.21: Annual Open Pit and Waste Rock Storage Facility Plan – End of Year 9

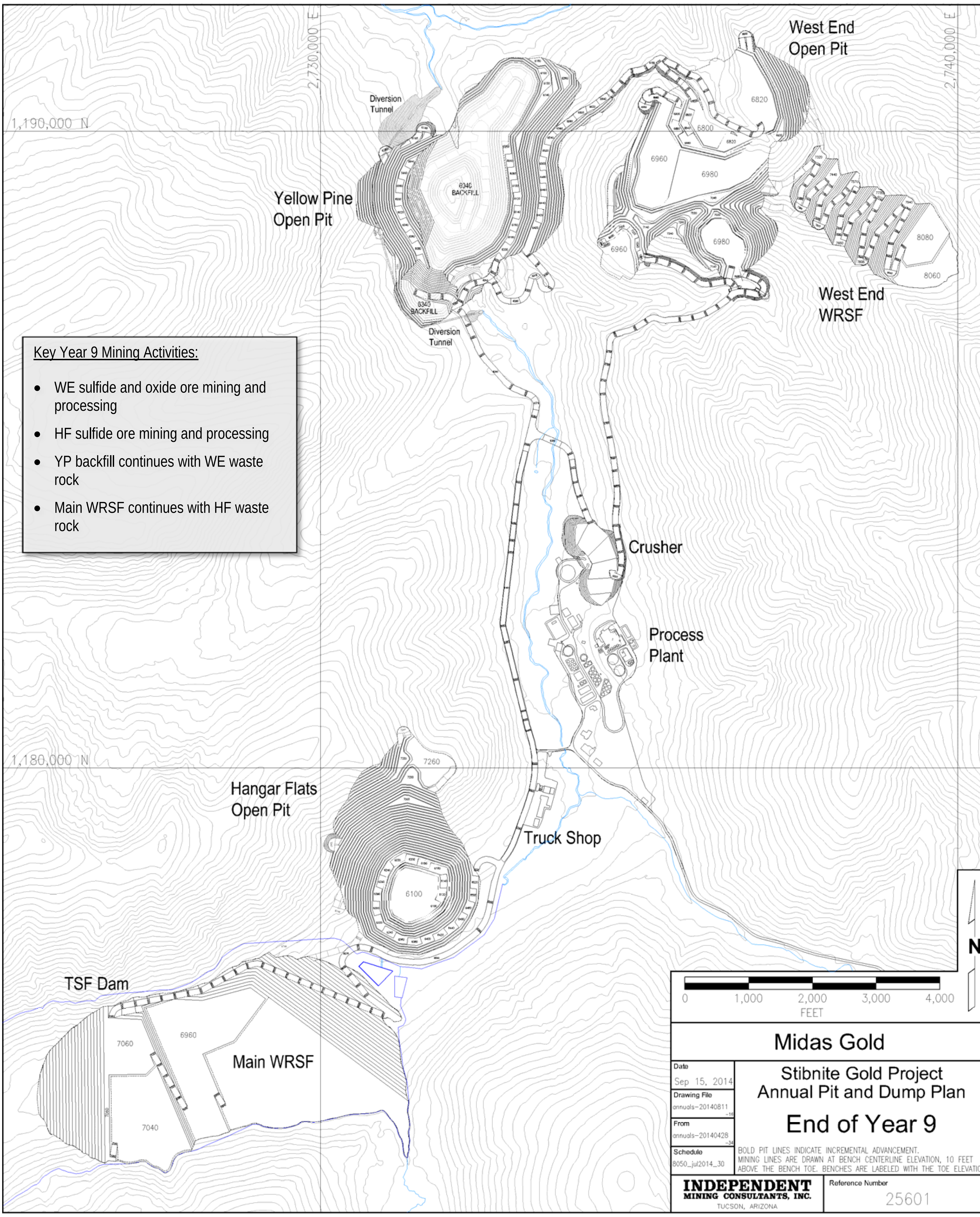


Figure 16.22: Annual Open Pit and Waste Rock Storage Facility Plan – End of Year 10

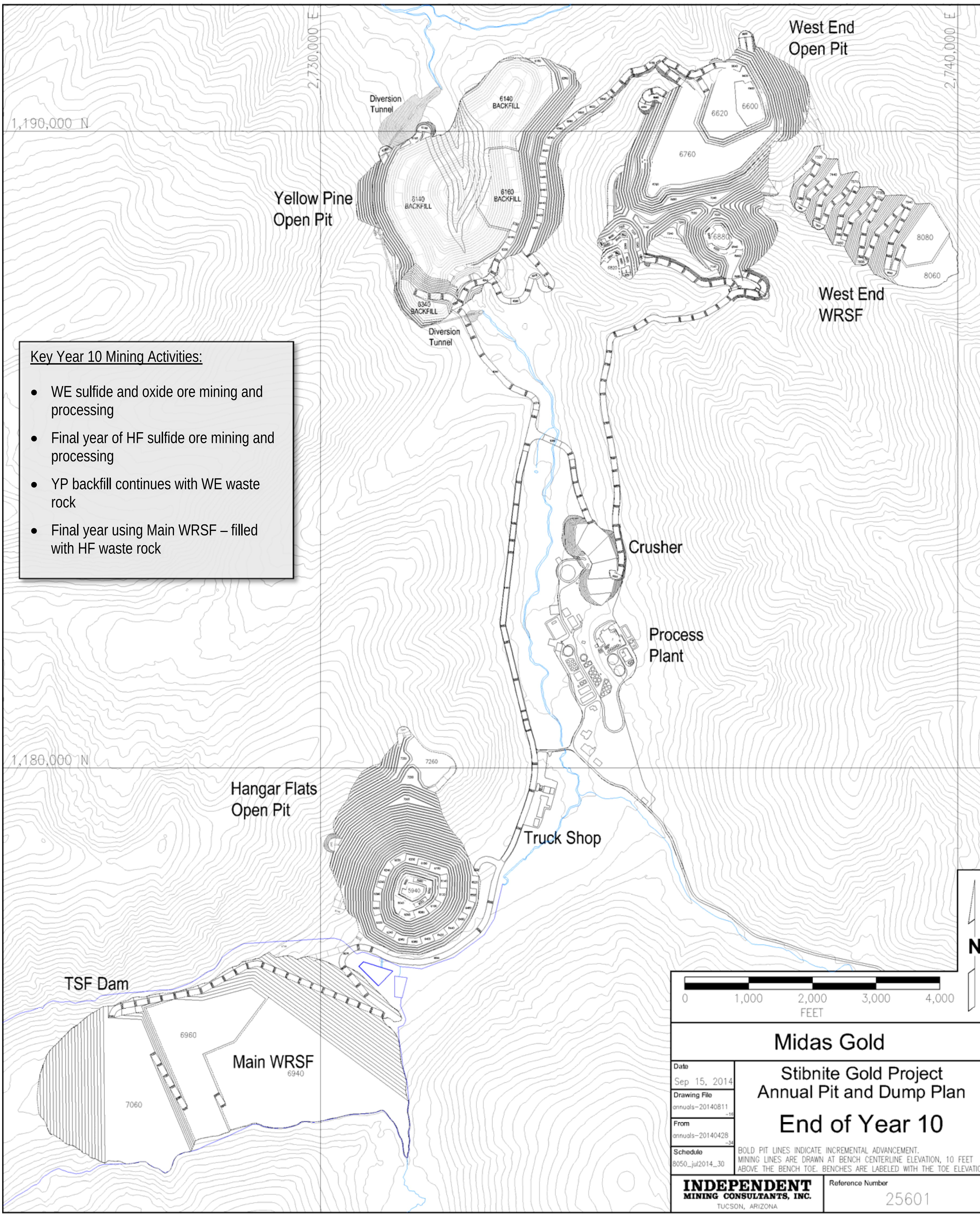


Figure 16.23: Annual Open Pit and Waste Rock Storage Facility Plan – End of Year 11

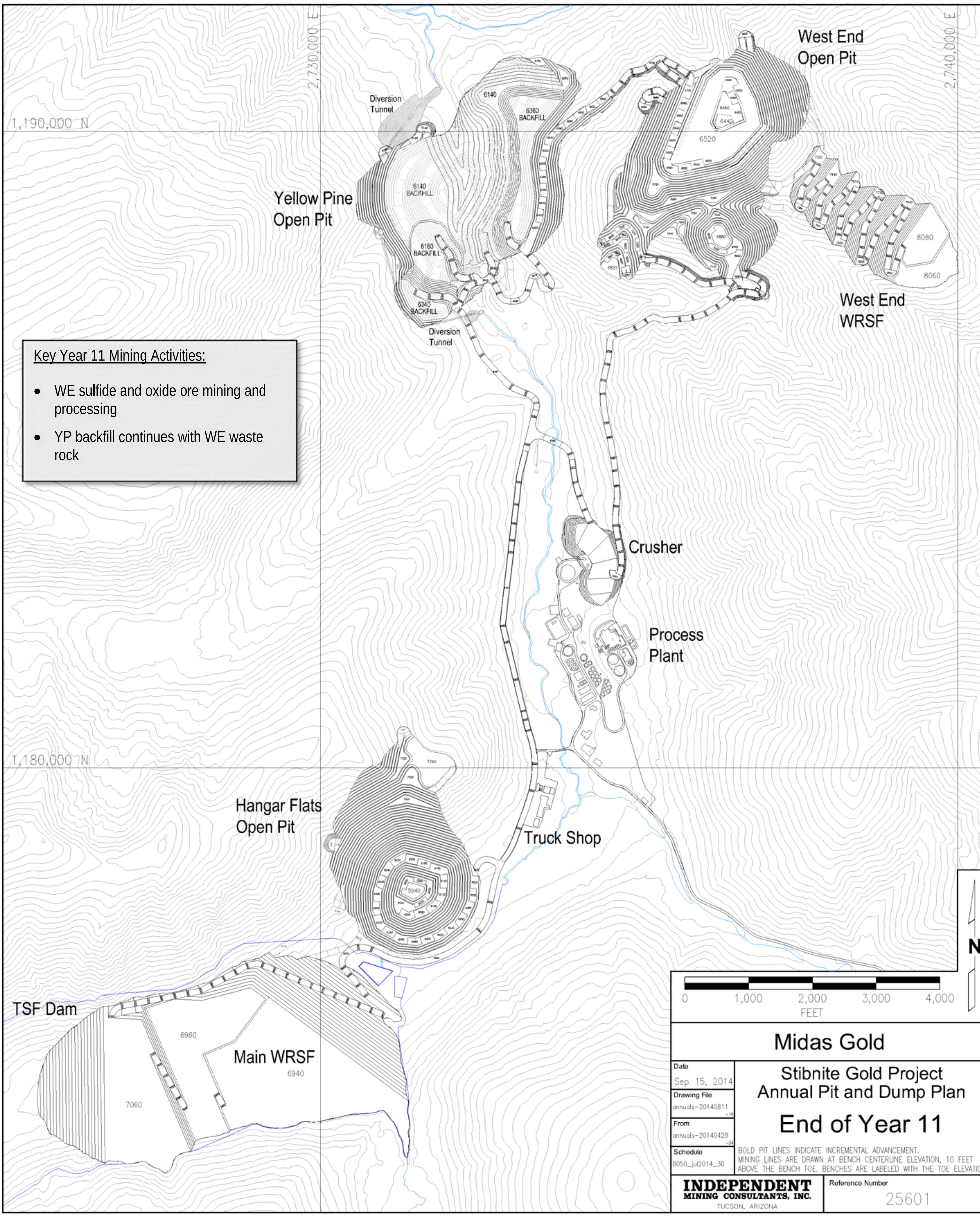
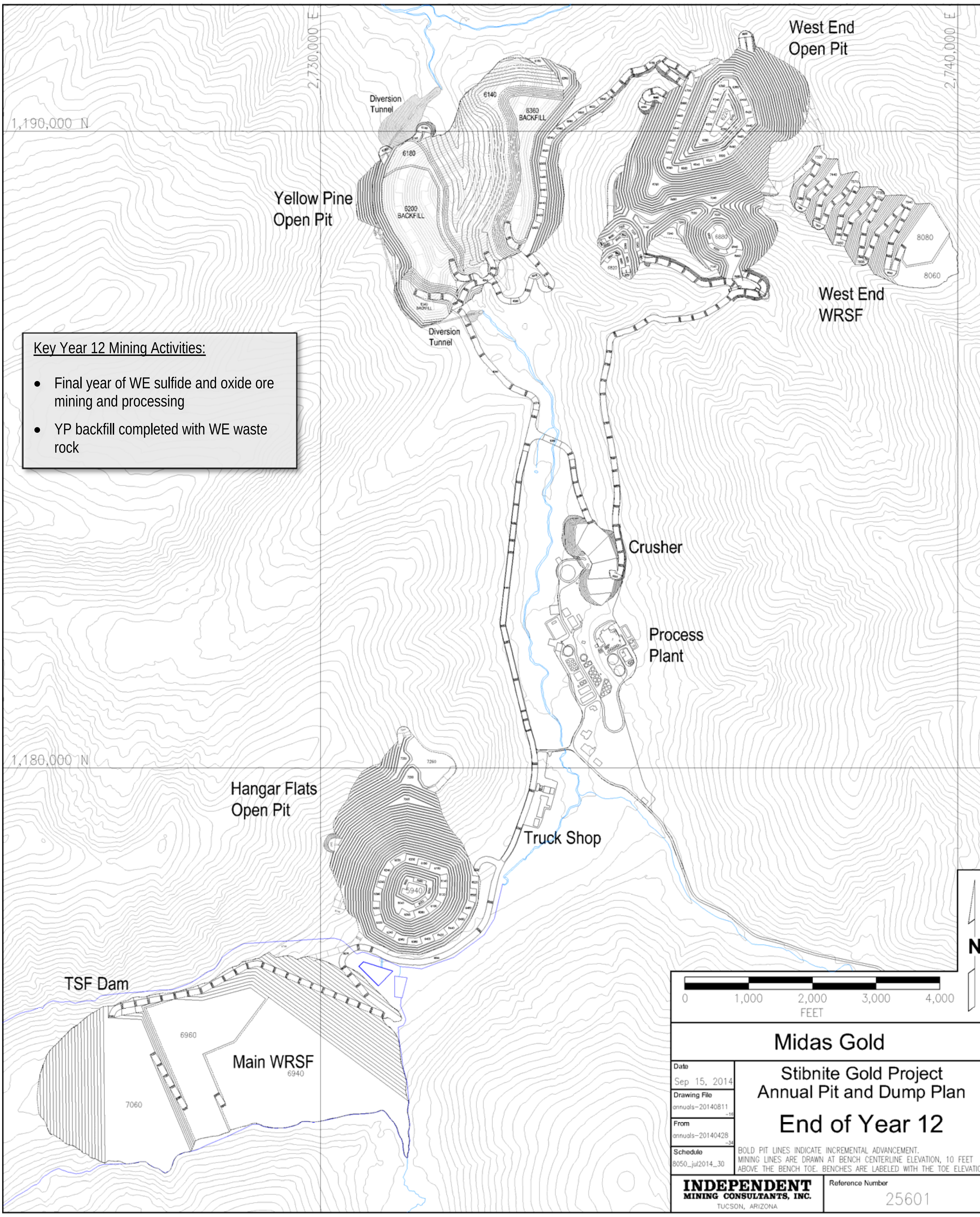


Figure 16.24: Annual Open Pit and Waste Rock Storage Facility Plan – End of Year 12



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17 RECOVERY METHODS

17.1 OVERVIEW

The Stibnite Gold Project process plant has been designed to process both sulfide and oxide mineralized material from three deposits (Hangar Flats, Yellow Pine, and West End) as well as Historic Tailings from former milling operations. The design of the processing facility was developed based on the laboratory testing, summarized in Section 13, to treat an average of 22,046 st/d, 365 days per year for a total of 8.05 million tons per year.

ROM material would be crushed and milled, then flotation and hydrometallurgical operations would be used to recover antimony as a stibnite flotation concentrate (with some silver and minor gold), doré bars containing gold and silver, and small quantities of elemental mercury, collected in flasks, to prevent its potential release into the environment. Historic Tailings would be introduced into the ball mill during the first 3 - 4 years of operation. Tailings from the operation would be deposited in a geomembrane-lined TSF. A simplified process flow diagram is shown on Figure 17.1 and a list of major equipment, including the estimated connected power requirements, is shown in Table 17.1.

Figure 17.1: Simplified Process Flow Diagram

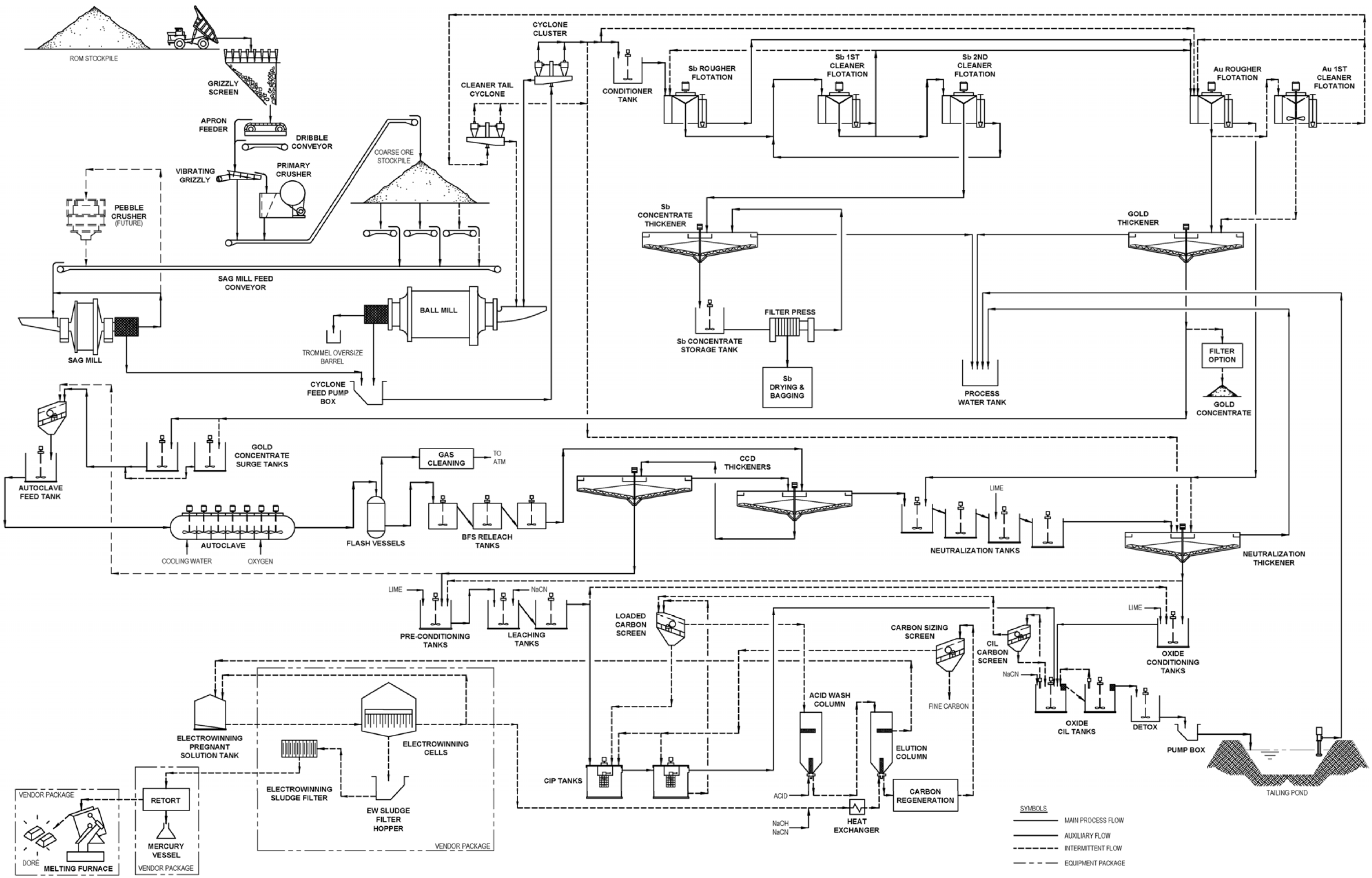

Table 17.1: Major Process Equipment List and Estimated Connected Power Requirements

No	Item	Description	Estimated Connected Power Required (kW) Each	Total
1	Primary Jaw Crusher	Metso C200 Jaw Crusher; feed opening 79 in x 60 in; 160 in wide x 264 in long x 111 in high	373.0	373.0
1	Cyclone Cluster	10 cyclones in cluster; gMax26 type		
1	Semi Autogenous Grinding (SAG) Mill	30 ft diameter x 16 ft effective grinding length	7,500	7,500
1	Ball Mill	24 ft diameter x 40 ft effective grinding length	13,500	13,500
1	Cyclone Overflow Analyzer	On-Stream Analyzer		
5	Sb Rougher Flotation Cell	2,500 ft^3 Tank Cell	90.0	450.0
6	Sb 1st Cleaner Flotation Cell	350 ft^3 Tank Cell	18.7	112.2
4	Sb 2nd Cleaner Flotation Cell	180 ft^3 Tank Cell	11.0	44.0
9	Gold Rougher Flotation Cell	17,650 ft^3 SuperCell or similar	447.6	4028.4
7	Gold 1st Cleaner – Cleaner Scavenger Flotation Cells	5,650 ft^3 tank cell	186.5	1305.5
1	Gold Concentrate Thickener	100 ft diameter high – rate thickener	15	15
1	Autoclave	15.1 ft ID x 106 ft t/t; hemispherical heads; brick lined, seven compartment, agitated		
7	Autoclave Agitators		112	784
2	Flash Vessels	15.5 ft diameter x 28 ft high, brick lined		
3	Basic Ferric Sulfate (BFS) Releach Tanks	29 ft diameter x 31 ft high; Super Duplex Steel; closed top; agitated	37.5	112.5
2	Countercurrent Decantation (CCD) Thickeners	170 ft diameter high rate thickener	15	30
6	Neutralization Tanks	52 ft dia. x 54 ft high, four tanks of 316L, two tanks of carbon steel; closed top, agitated	75	450
1	Neutralization Thickener	150 ft diameter high-rate thickener	15	15
2	Concentrate Preconditioning Tanks	49 ft diameter x 51 ft high; carbon steel, agitated	18.7	37.4
4	Concentrate Leaching Tanks	52 ft diameter x 54 ft high; carbon steel, agitated	37.5	150
6	Concentrate CIP Tanks	20 ft diameter x 30 ft tank height; 2 ft freeboard; carbon steel with pump cells	37.5	375
2	Detox Tanks	40 ft diameter x 42 ft high	112	224
2	Oxide Conditioning Tanks	54 ft diameter x 56 ft height	112	224
6	Oxide CIL Tanks	54 ft diameter x 56 ft height	112	672
1	Carbon Regeneration Kiln	500 lbs/hr carbon throughput; electric fired; 1,290 F (design temp); 10 min retention at temp	11.2	11.2
1	Elution Vessel	4 to 1 height to diameter ratio; CS; 300º F (design temp); propane heater		
1	Fresh/Fire Water Tank	40.0 ft diameter x 42.0 ft high		
2	Lime Silo	54,000 ft^3 bolted tank; 30 ft diameter x 76 ft cylinder height 60º cone bottom		
2	Lime Slaker Plant	Vulcan DV-225; 9 st/h detention-type lime slaker system	164.1	328.2
1	Oxygen Plant	27.8 st/h @ 95% purity; 82.4º F; 570 psig	13,000	13,000

17.2 MINE PRODUCTION SCHEDULE SUMMARY

A preliminary mine schedule, listing elemental concentrations of interest needed to drive the process design, is shown in Table 17.2. The data in Table 17.2 does not represent the final PFS mine schedule, as the final information was not available early in the process design studies; however, the elemental trends closely align with the final PFS data. Review of Table 17.2 indicates that the gold, sulfur, and calcium concentrations within the Project deposits are highly variable. The material to be processed early in the life of the operation is relatively high in gold and sulfur concentration; however, after year four, the blend trends toward lower gold and sulfur but higher calcium concentrations. These changes have important implications for the process plant design. In addition to the higher grade, freshly mined material from Yellow Pine, Historic Tailings would be added to the process at 10 - 15% of the total throughput during the first four years of the operation. The Historic Tailings are expected to average 0.03 oz/st gold and 0.4% sulfur, with a typical size of 80% passing 180 microns.

Table 17.2: Primary Crusher Feed Schedule with Process Elements of Interest

Time Period	Ore (kst)	Au (oz/st)	Sb (%)	Sulfur (%)	Calcium (%)
-1	508	0.076	0.279	1.28	1.06
1	5,072	0.066	0.119	1.31	1.08
2	8,050	0.055	0.068	1.17	1.32
3	8,051	0.054	0.081	1.09	1.26
4	8,050	0.058	0.072	1.14	1.34
5	8,049	0.051	0.038	0.87	1.05
6	8,051	0.051	0.027	0.78	1.80
7	8,049	0.033	0.020	0.46	2.96
8	8,049	0.042	0.047	0.53	4.15
9	8,050	0.046	0.107	0.71	3.47
10	8,051	0.036	0.034	0.59	3.37
11	8,050	0.036	0.015	0.61	3.30
12	5,444	0.044	0.005	0.65	4.82
Total / Average	91,524	0.047	0.053	0.82	2.46

On average, 12% of the material shown in Table 17.2 would be processed through the antimony recovery circuit, with annual values ranging from approximately 27% in year one to less than 2% in year 12. Approximately 13.5% of the material noted in Table 17.2 is oxide and responds well to conventional cyanidation, but poorly to flotation. An additional 12% of the material noted in Table 17.2 is characterized as transition material and yields variable gold recoveries by both flotation and conventional cyanidation. The remaining 74.5% of the material noted in Table 17.2 is considered refractory to direct leaching to recover gold and silver but responds well to flotation to a concentrate.

17.3 PROCESS DESCRIPTION

The flow sheets developed for the Stibnite Gold Project PFS are based on metallurgical test programs directed and supervised by Blue Coast Metallurgy (BCM); the metallurgical testing was primarily conducted by SGS Minerals Inc. (SGS). Previous testing to support the PEA was also supervised by BCM and conducted by SGS.

The process plant was designed to process 22,046 st/d through crushing, milling/grinding, flotation and tailings processing operations. Zones in both Yellow Pine (YP) and Hangar Flats (HF) contain sufficient antimony to warrant processing for antimony recovery. The antimony would be recovered as stibnite flotation concentrate and would be shipped off-site for further processing.

Metallurgical testing indicates that the refractory sulfides containing gold can be recovered to a flotation concentrate. The gold can then be liberated by oxidation of the sulfide minerals and recovered by cyanide leaching of the oxidation residue. Fully- and partially-oxidized material, also referred to as oxide and transition material, respectively, yield less consistent flotation recoveries; to improve metallurgical recoveries of gold and silver from the oxide and transition materials, the oxide and flotation tailings would be processed through a carbon-in-leach (CIL) process to recover cyanide-soluble gold not recovered in the flotation step. The gold produced as doré bars at site and containing gold and silver would be sold to third parties for further processing. Minor amounts of mercury are also present in the material to be processed; equipment would be installed to recover the mercury that is with the gold, transport it to a permitted off-site facility, prevent its discharge into the environment, and maintain a safe working environment for employees.

Process design criteria were developed for each process area. Data used in the process design criteria are from various sources including:

1) PEA (SRK, 2012);
2) client provided historical data conducted by and for prior owners and operators of the Project;
3) metallurgical testing;
4) calculations;
5) vendor data or recommendations;
6) M3 database information;
7) industry practice;
8) handbooks;
9) assumptions based on experience; and
10) other reports and consultants.

The following sections provide a comprehensive summary of: the PFS process flowsheet based on the metallurgical testing and interpretation presented in Section 13; the major process equipment selected for the Project and a discussion of the alternatives considered; a description of the primary buildings required to support the major process equipment; and descriptions of the primary process support infrastructure including the water systems, process air systems and the tailings handling system. The layout of the facilities discussed in this section, and of the alternative layouts considered, is discussed in Section 18.

17.3.1 Crushing Circuit

ROM material would be delivered by mine haul truck to the primary crusher, or to one of four 100,000 ton capacity ROM stockpiles. The stockpiles provide surge for the high-antimony and oxide materials that can be campaigned through the process plant, and allow surge for blending of material for control of sulfide and carbonate concentrations.

The crushing circuit design was developed based on a 24 hour per day, 365 day per year operation at an average utilization of 75% yielding an instantaneous design-throughput of 1,225 st/h. ROM material would be dumped onto a grizzly screen and into the crusher dump hopper. A front end loader would be used to feed stockpiled material to the crusher as needed for blending. The dump hopper would have live capacity for one dump truck. A rock breaker would be installed at the dump pocket to handle oversize. An apron feeder would draw material from the dump hopper to feed a vibrating grizzly and grizzly oversize would feed the jaw crusher.

A trade-off study was completed to evaluate the economics and operational flexibility of various crushing and grinding options. The mill feed from the WE pit requires more energy for crushing and SAG grinding than the YP and HF mill feed. Crushing options evaluated included one jaw crusher, two jaw crushers, or a gyratory crusher. Grinding options included a single large SAG mill – ball mill circuit, one smaller SAG mill with pre-crush of harder WE material and conventional three-stage crushing. All of the combinations were evaluated in terms of projected capital and operating costs using a net present cost analysis with a 5% discount rate. The analysis indicated that a single large jaw crusher with a large SAG mill installed from the beginning of the operation had the lowest net present cost and requires no additional construction in the comminution circuit later in the mine life, which could be disruptive to the operation. It also has the benefit of enabling higher production rates in the early stages of mine life to shorten the payback period. Blending of HF and WE material was recommended to control hardness variations.

A large jaw crusher was selected for the Stibnite Gold Project since both YP and HF ROM material are expected to contain a high percentage of fines and, as noted in Section 13, both have a relatively low crushing work index; the WE work index is characterized as average for gold deposits globally. Crushing simulations with vendor supplied software support the selection of a jaw crusher for the Project.

The primary crusher would be installed in a concrete and steel building with a 100 ft long x 40 ft wide x 128 ft high concrete dump pocket. The steel structure would be supported on concrete piers and include preformed insulated metal roof and wall panels; a 20-ton overhead bridge crane would also be included. The ROM material would pass two vibrating grizzlies then report to the primary jaw crusher; the crusher discharge and grizzly undersize would be transferred via conveyor to the coarse ore stockpile through a reinforced concrete tunnel. The crusher production rate would be monitored by belt scale and tramp iron would be removed using a magnet. A metal detector would also be installed on the stockpile feed conveyor. Water sprays would be installed at the crusher dump pocket and at material transfer points to reduce dust emissions.

The stockpile was designed to have a 12-hour live capacity, with approximately 33,000 st (1.5 days) of total capacity. Three feeders would be provided for material reclaim to the milling circuit. The stockpile would be covered to reduce dust emissions and to protection the material from inclement weather. A dust collector would be installed to control dust in the reclaim tunnel.

The crushed-ore stockpile building was designed as a domed structure with a 240 ft inside diameter at the concrete ring dome spring line. The concrete ring would be supported by 24 concrete piers, 18 ft-6 inches high, arrayed about the center of the dome on 15^{o} angles. The dome rises 92 ft-9 inches above the concrete ring and is comprised of coated metal tube framing with metal roof/siding attached to the metal framing. There would be four solid concrete 15^{o} segments evenly spaced around the perimeter of the dome for lateral purposes. The crushed ore stockpile would be reclaimed through a 20 x 20 x 160 ft concrete reclaim tunnel through two of the three draw-holes and belt-feeders; the belt feeders would transfer the material to the SAG mill feed conveyor, which transfers the crushed ore to the grinding circuit.

17.3.2 Grinding Circuit

The grinding circuit design was developed based on a 24 hour per day, 365 days per year operation with an average utilization of 92% yielding an instantaneous design-throughput of 998.5 st/h.

Reclaimed material, recycled pebbles, reagents and process water would be fed to the SAG mill circuit, and the SAG mill discharge would be screened and the screen undersize discharged to the grinding sump; screen oversize would be recycled to the SAG feed system. Grinding test work completed to date indicates that a recycle (pebble) crusher is not required for efficient processing during the early years of operation, but recycle pebble crushing may improve grinding circuit performance in the later years of operation, depending on the blend of HF and WE material.

The SAG screen undersize would be combined with the discharge from the ball mill in the cyclone feed pump box, then pumped to a cyclone cluster for classification. When Historic Tailings are processed during early years of the operation, the slurry from the plant would also flow to the cyclone feed pump box. Cyclone underflow flows by gravity to the ball mill; cyclone overflow, at 33% solids with a target size of 80% passing (P_{80}) 75 microns, would be screened to remove tramp oversize and flow through a feed sample system and on to the antimony or gold rougher flotation circuit, depending on the antimony concentration of the material.

The grinding circuit was designed to include one 30-ft diameter by 16-ft effective grinding length (EGL), 7,500 kW SAG mill and one 24-ft diameter by 40-ft EGL, 13,500 kW ball mill, based on results from JKSimMet simulations using the 75th percentile hardness data for the grind characteristics for each material type described in Section 13. The equipment is large, but considered proven in the industry.

The grinding building was designed as an enclosed steel and concrete building approximately 160 ft wide x 220 ft long x up to 140 ft high (at the ridge). The steel structure is supported on concrete piers and supports preformed insulated metal roof and wall panels. An on-stream analyzer that can provide metal and sulfur analysis would be included for circuit control. The grinding area floor would be concrete on grade with containment walls to contain spills within the floor area. The floor would be sloped to a trench that directs spillage to a sump that would pump the contained material back to the mill feed. A bridge crane would be provided to service the mill area and SAG and ball mill liner handlers are provided to facilitate mill liner maintenance.

17.3.3 Flotation Circuit

The flotation circuit consists of up to two sequential flotation stages to produce two different concentrates; the first stage of the circuit was designed to produce an antimony-rich concentrate, and the second stage was designed to produce a gold-rich concentrate. If the antimony content of the feed material is not in economic concentrations then the antimony circuit would be bypassed and a gold bearing sulfide concentrate would be the only concentrate produced by the flotation circuit.

17.3.3.1 Antimony Flotation

The test data used for the material balances to size the antimony flotation circuit are from the YP-high antimony and the HF-high antimony locked cycle test results described in Section 13.

Reagents are added to grinding to depress gold bearing sulfides prior to stibnite (antimony sulfide) flotation. Discharge from the grinding circuit flows to the antimony rougher conditioning tank where lead nitrate solution is added to activate the stibnite.

The conditioned pulp reports to the antimony rougher flotation bank where other flotation reagents are added as needed. The antimony rougher flotation circuit was designed to recover the stibnite into the rougher concentrate; the objective is for gold bearing sulfides not to be recovered at this point of the circuit. Antimony rougher tailings would be combined with antimony first cleaner tailings and pumped to the gold rougher conditioning tanks. The antimony rougher operation includes one bank of five 2,500 ft^3 flotation cells with a total retention time of seven minutes. The plant cell selection was made considering a balance with the number of flotation cells in series to reduce the impact of short-circuiting, the maximum flow recommended for the flotation cells, and the desire to minimize the gold bearing sulfide flotation to the antimony concentrate.

Antimony rougher concentrate would be pumped to the antimony first cleaner conditioning tank where reagents could be added, as required, and the rougher concentrate would be mixed with antimony second cleaner tailings and discharge by gravity to the antimony first cleaner flotation. The antimony first cleaner operation includes one bank of six 350 ft^3 flotation cells with a total retention time of seven minutes.

Antimony first cleaner concentrate is pumped to the antimony second cleaner conditioning tank where it is conditioned with reagents as needed prior to antimony second cleaner flotation. Antimony first cleaner tailings would be combined with antimony rougher tailings to feed the gold rougher conditioning tanks. Antimony second cleaner tailings would be pumped to the first cleaner conditioning. The antimony second cleaner operation includes one bank of four 180 ft^3 flotation cells with a total retention time of seven minutes.

The second cleaner concentrate is the final antimony concentrate and would be sampled, thickened, filtered, dried, stored and bagged for shipment. The antimony thickener was sized at 25-ft diameter based on thickening test results indicating a unit rate of 1.46 ft^2 per ton per day of concentrate. The antimony concentrate filter and dryer were sized based on general vendor guidelines for similar material. Dried concentrate would be stored in a bin prior to bagging for shipment.

17.3.3.2 Gold Flotation

When low-antimony sulfide ore is processed by the grinding circuit, the ball mill cyclone overflow bypasses the antimony flotation circuit and feeds the gold rougher conditioning tank; when high-antimony sulfide ore is processing by the grinding circuit, antimony rougher tailings feeds the gold rougher conditioning tank. In the gold rougher conditioning tank, copper sulfate solution would be added to activate the sulfides; the conditioned pulp would discharge to the gold rougher flotation bank where additional flotation reagents would be added, as needed. The gold rougher flotation circuit was designed to recover the gold-bearing sulfides into the rougher concentrate; gold rougher tailings would flow to the neutralization tanks or the neutralization thickener. The gold rougher operation includes one bank of nine 17,650-ft^3 flotation cells with a total retention time of 80 to 90 minutes. The plant cell selection was made targeting plant to lab retention time factor of 2.5 to 3.0 and considering a balance with the number of flotation cells in series to reduce the impact of short-circuiting, and the maximum sized flotation cell currently being manufactured.

Generally, when processing YP and HF material, the rougher concentrate grade is suitable for processing through the oxidation circuit directly and would advance to the gold concentrate thickener. Some YP and HF zones, and WE rougher concentrate, would need to be cleaned to reject carbonates or improve the concentrate sulfur grade for efficient processing through the oxidation circuit. If advantageous, rougher concentrate would be pumped to the gold first cleaner conditioning tank where flotation reagents could be added, if needed, and flow by gravity to the gold first cleaner flotation followed by the gold cleaner scavenger flotation. The gold cleaner and cleaner-scavenger operation includes one bank of seven 5,646-ft^3 flotation cells with a total retention time of 75 minutes, which is approximately 2.5 times the laboratory retention time for the combined cleaner and cleaner scavenger operation.

The combined gold cleaner and gold cleaner scavenger concentrate or the gold rougher concentrate, when grade is high enough, flow to the concentrate sampler and discharge to the gold thickener. The thickener serves to adjust the pulp percent solids prior to the oxidation step for efficient pulp storage and to facilitate autoclave temperature control. Thickener overflow is returned to the process water system.

To size the concentrate thickening, storage and all downstream concentrate operations, a maximum mass pull of 20% of the rougher feed tonnage was assumed. In practice, if this mass pull could not be achieved and a concentrate grade of 5 to 6% sulfide sulfur maintained, then the mass pull would be reduced to maintain the target sulfide sulfur concentration in the autoclave feed. The target sulfide sulfur grade range of 5 to 6% was set to target auto-thermic autoclave operation and was determined based on experience at other operations.

The gold cleaner scavenger tailings would be sent through a cyclone with the coarse fraction being recycled to the primary milling circuit for additional grinding when processing low antimony material and the fine cyclone overflow reporting with the cyclone overflow to the trash screens.

A 200 ft long x 70 ft wide x up to 140 ft high building was designed to house both stages of the flotation circuit. The structure would be supported on concrete piers that support preformed insulated metal roof and wall panels. Two 20-ton overhead bridge cranes, one for each side of the building, are planned. In addition to housing the antimony and gold flotation cells, the structure supports the antimony concentrate thickening, and the pressing and drying facilities.

17.3.4 Gold Flotation Concentrate Oxidation Circuit

The primary product from the gold flotation circuit is an auriferous pyrite concentrate; arsenopyrite and arsenian pyrite are also present in the concentrate. In order to liberate finely encapsulated gold particles in the concentrate, it must be oxidized. The products of oxidation are generally ferric arsenate (scorodite) and sulfuric acid; liberated gold and silver are present within the solids.

17.3.4.1 Oxidation Circuit Trade-Off Study

Four methodologies were considered viable technologies for oxidizing the gold concentrate: pressure oxidation using an autoclave under high oxygen pressure (POX); biological oxidation (BIOX) using bacteria in reactor tanks via the proprietary BIOX® technology of Biomin South Africa Pty (Biomin); Xstrata Plc's Albion Process (Albion) and roasting. Roasting and Albion were evaluated using historically available test work from the Project, and generic results available from other projects, respectively; no testwork was completed for these methodologies as part of the PEA or PFS given the anticipated complexities and costs for these processes. POX and BIOX were evaluated with the support of a comprehensive metallurgical testing program for each technology and a technical trade-off study that was developed with supporting design and cost information from M3 and Biomin.

The POX/BIOX trade-off study included detailed metallurgical recovery estimates based on test work from all three deposits that make-up the Stibnite Gold Project, capital and operating costs estimates, environmental and closure considerations, technical risk, permitability and other considerations. Design criteria were compiled for the POX option by M3 while Biomin provided the design criteria, equipment list, and pricing for the BIOX option. M3 prepared flowsheets for the POX and BIOX options so that equipment lists could be compiled. Independent pricing was solicited for POX and BIOX capital equipment. General arrangement drawings were prepared for each option so that material take-offs for foundations and structural steel.

The trade-off study results indicate that the POX option is the most economical alternative, has lower technical risk, more certain and improved environmental outcomes, and has been permitted in the US for a number of gold operations. Table 17.3 summarizes the estimated range of potential capital costs, operating costs and life-of-mine unit operating costs for POX and BIOX. The numbers are comparative; consequently, contingency was not included or deemed necessary in the estimates.

Table 17.3: Estimated LOM Capital and Operating Cost Summary for POX and BIOX

Oxidation Option	Capital Cost	LOM Operating Cost	Operating Cost per Ton of Concentrate
POX	$93,948,000	$476,809,000	$52.57
BIOX[1]	$85,413,000	$784,379,000	$86.48
BIOX[2]	$144,258,000	$784,379,000	$86.48
BIOX[3]	$163,215,000	$784,379,000	$86.48

Notes:
(1) Biomin CAPEX less CCD and neutralization section, Biomin equipment pricing, Biomin construction factors.
(2) M3 CAPEX less CCD and neutralization section, Biomin equipment pricing, M3 take-off & construction factors.
(3) M3 CAPEX less CCD and neutralization section, M3 equipment pricing, M3 take-off & construction factors.

Biomin's capital cost estimate for the BIOX option is appreciably lower than M3's engineering build-up; whether the CAPEX lies closer to Biomin's estimate or M3's, the operating cost is significantly lower for pyrite oxidation using POX versus BIOX. The operating cost for the BIOX process is higher due to higher cyanide consumption and higher limestone consumption. The BIOX cyanide consumption is higher due to the reaction of cyanide and reduced sulfur species that form thiocyanates. In the POX process, the sulfur is primarily oxidized to sulfate and so the formation of thiocyanate is much lower. The limestone consumption for BIOX is higher since it is necessary to add limestone to the BIOX reactor to control the acid concentration during the BIOX reaction. For the POX process, limestone is not required since most of the acid generated during the oxidation is neutralized by the flotation tailings.

It may be possible to lower the BIOX operating costs by using onsite mining of limestone for pH control during oxidation within the reactors; however, analysis indicates this improvement only partially closes the gap and is at the expense of requiring a limestone process facility including plant operators, extra power, spare parts and maintenance.

The POX alternative provides a robust process to oxidize concentrate with a retention time in the autoclave of one hour compared to five days in the BIOX reactors. The risk for prolonged downtime for a technical issue with the oxidation circuit is higher for POX than for the BIOX equipment, which by its nature is simpler; however, the delicate conditions required to keep the bacterial culture alive and at peak efficiency has been a risk experienced with previous bio-oxidation operations and offsets this advantage.

The other risk factor for the two processes is that of gold recovery. Pressure oxidation is a high-energy oxidation that reaches nearly complete oxidation of sulfides. The BIOX process works on a bench scale but, in practice, with concentrate spread between 48 large reactor tanks, the opportunity for short circuiting and scale-up inefficiencies to produce lower-than-expected recoveries is a risk that has to be considered and has been experienced at operations in other locations. In order to determine risks, pilot testing of the BIOX process or evaluation of data from operating plants would have to be conducted to determine if the batch test oxidation recoveries hold up on a larger continuous scale; there is risk that bio-oxidation recoveries would diminish in practice. While one of the recommendations in this PFS is for pilot testing the POX process, this is not related to recoveries, where the risk is seen as low based on the extensive test work completed to date with consistent results, rather it is intended to provide additional information for environmental and neutralization conditions.

17.3.4.2 Pressure Oxidation Circuit

Two concentrate surge tanks provide approximately 16 hours of live surge and blending as a buffer between the circuits. Discharge from the surge tanks is pumped through a trash screen to the autoclave feed tank, located near the autoclave. The autoclave feed tank provides about one hour of live surge near the autoclave and allows the operator better control of the autoclave feed. When processing cleaner concentrates, the autoclave feed tank would also blend second stage counter-current decantation (CCD) thickener underflow with the cleaner concentrate to reduce the sulfide concentration to the target range of five to six percent. The recycle would allow the autoclave to run at a higher percentage of solids concentration and reduce problems with scale formation.

Two independent trains of two pumps in series are provided to feed the autoclave. The first stage of each train is a centrifugal booster pump; the second stage of each train is a positive displacement pump. During normal operation, both pump lines would be in operation. When one pump line is down for maintenance, the second pump line would continue to operate with a maximum capacity of 75 to 80% of required volume. During this time, the gold concentrate surge tank would contain the surplus flow.

The autoclave would normally operate at 428˚F and 425 psig. The autoclave is designed to oxidize to sulfate 12.7 st/h of sulfide sulfur. Concentrate would be pumped into the first and largest compartment of the autoclave, containing four agitators. Pulp discharge from the first compartment flows through the remaining three compartments in series, each with one agitator. The nominal retention time of the autoclave is 60 minutes and the design sulfide sulfur oxidation is +99%. The estimated oxygen utilization is 90% due to the relatively low carbonate concentration of the feed. The target autoclave feed sulfide sulfur grade is 5.0 to 6.0% in order to achieve autothermic conditions and avoid the use of cooling water. The gold flotation concentrate has been thickened to approximately 50% solids prior to the autoclave so the autoclave feed percent solids can be controlled to facilitate autoclave temperature control. Cooling water can also be pumped to the autoclave as needed for the final temperature control. A vendor package plant would provide 670 st/d of oxygen gas at 95% purity. Steam generators are provided for initial autoclave heat up and to add heat to the process during upset conditions.

Sizing of the autoclave is based primarily on the rate that sulfide sulfur is fed to the autoclave. For the PFS design, the autoclave design is based on a sulfide sulfur feed rate of 12.7 st/h. This value is approximately 30% higher than the PEA design value. The relatively high sulfide sulfur feed rate for the PFS is a result of the sulfur feed grade variation shown in Table 17.2. As noted in Table 17.2, the higher grade gold is also higher in sulfide sulfur and the autoclave must be designed to process this high sulfur material. The autoclave is designed to operate at 5 - 6% sulfide sulfur concentration and 50% solids by weight. The relatively high percent solids in the autoclave feed does not significantly increase the autoclave size, but provides the benefits of an autothermic operation at lower sulfide sulfur concentration and reduced scaling due to the high solids content of the feed. At the stoichiometric requirement of 1.87 tons of oxygen per ton of sulfide sulfur, and estimated 90% oxygen utilization, the oxygen requirement is 27.9 st/h of gas at 95% oxygen for the design throughput. Design conditions of one hour of retention time and cooling water addition for process control result in a live reactor volume of approximately 15,700 ft^3. The total reactor volume allowing for head space and an estimated operating level of 83% results in a total volume of 18,900 ft^3. The resulting internal dimensions of the autoclave are 15.1 ft diameter (inside brick) and 106 ft in length (tangent to tangent) with an overall length of 114.3 ft.

Autoclave discharge would flow through two flash vessels in parallel. Slurry discharge from the flash vessels would flow by gravity to the basic ferric sulfate (BFS) re-leach tanks. The BFS re-leach is required to dissolve basic ferric sulfate [$FeSO_4(OH)$] precipitated during the autoclave operation. The three tanks in series would provide a total retention time of approximately 6 hours for this operation. Flash vessel gas phase discharge would be scrubbed in a single stage venturi scrubber and discharge to the atmosphere. Depending on operating conditions, approximately 15%, or 50 st/h, of the autoclave feed moisture would be lost in this stream as steam.

The gold concentrate surge tanks were designed to be 49 ft diameter x 51 ft tall, carbon steel tanks that feed the autoclave feed tank that feeds the autoclave housed in the autoclave building. This structure is an L-shaped steel and concrete structure supported on concrete piers and supports preformed insulated metal roof and wall panels. There is one 10-ton overhead bridge crane. One branch of the L-shaped building is approximately 180 ft long x 60 ft wide x 67 ft high (at the peak). It houses the autoclave and supporting tanks and vessels. The other leg is 80 ft long x 60 ft wide x 30 ft high; this wing houses the site assay lab, the steam plant, and an electrical room.

The slurry from the autoclave flows to the two exterior mounted flash vessels, and from there to the BFS re-leach tanks. After stepping through each of the three tanks, 29 ft diameter x 31 ft high, the slurry flows by gravity to the CCD thickeners. The average annual pressure oxidation circuit utilization was estimated to be 85%.

17.3.5 Pressure Oxidation Products Handling

Acid and soluble salts produced during the oxidation process would be separated from the solids in the CCD circuit. A two-stage CCD circuit with a wash ratio of 6:1 is planned. The wash water supplied to the CCD system would be process water with a neutral pH. The CCD thickeners would be high-rate thickeners. CCD thickener underflow would advance by pumping from thickener to thickener; CCD thickener overflow would be cooled in spray towers and advance to neutralization. Two cooling towers are required to provide the cooling. Since the solutions treated are nearly saturated, solids would precipitate during the cooling process. A spare cooling tower is installed to allow shutdown on a regular basis for routine cleaning and maintenance. During operation with high sulfur feed to the autoclave, approximately 50 st/h of water would be evaporated in the cooling tower operation.

To neutralize the CCD overflow, gold flotation rougher tailings would be mixed with the cooled CCD solution in six neutralization tanks arranged in series. Each tank is designed with a 1-hour retention time with full solution and tailings flow. The carbonates in the flotation tailings would react with the acid to precipitate metal sulfates and hydroxides in the first two stages. The quantity of tailings added to the CCD solution can be adjusted so the pH can be controlled at 2.5 in the first two stages, 5.0 in the third and fourth stages, and 7.5 to 8.0 in the last two stages. Depending on the feed blend and the sulfide and carbonate concentrations, materials treated early in the life of the operation may generate more acid than can be neutralized by the carbonate in the tailings. During these periods, lime would be added to the third tank to adjust the pH to 5.0. To complete the neutralization process lime would also be added to the fifth tank to adjust the pH to 7.0 to 8.0 prior to the neutralization thickener. Air would be sparged to the neutralization tanks to facilitate removal of carbon dioxide gas evolved during the reactions. The tanks would be covered and the discharge gas would be routed to a scrubber.

Slurry discharge from the neutralization tanks would be thickened in a high-rate thickener for water recovery. Thickener overflow would be pumped to the process water tank. Thickener underflow would flow to the tailings sump and be pumped to the TSF or the flotation tailings leach circuit, depending on the leachable gold concentration of the neutralization thickener underflow.

The CCD area consists of two high rate thickeners 170 ft in diameter in a concrete containment area. The overflow of the CCD thickeners goes through three cooling towers on its way to the neutralization tanks and then to the neutralization thickener. The neutralization tanks are six tanks 48 ft in diameter x 50 ft high. The first two tanks are open-topped and the other four are closed-top; all are agitated and they are located in a concrete containment area. The neutralization thickener is 136.6 ft in diameter with a 10-ft sidewall in a concrete containment area. Underflow from the neutralization thickener reports to the CIL tanks or to the tailings pump building. The underflow from the CCD thickeners reports to the pre-conditioning and leaching tanks.

17.3.6 Concentrate Leach and Carbon Handling

The second stage CCD thickener underflow would be pumped to two 49 ft diameter x 51 ft high pre-conditioning tanks; the slurry would be neutralized with lime with a total retention time of 10 - 12 hours. Slaked lime would be added to adjust the pH to 10.5 - 11.0 prior to cyanidation. The pre-conditioning tanks would overflow by gravity to four 52 ft diameter x 54 ft high leach tanks in series that would provide a total of 24 hours of retention time for the concentrate leach step. The tanks are designed to be stepped down, promoting gravity flow through each of the tanks.

Cyanide solution would be added to the first leach tank and air would be added to each tank to facilitate gold leaching. After leaching, the pulp would flow by gravity to the six 20 ft diameter x 32 ft high CIP tanks. A Kemix pumpcell CIP system is planned. The pumpcell system was selected to minimize gold inventory considering the high-grade concentrate leach system. Leached pulp would flow by gravity to the pumpcell feed launder. The feed launder valve arrangement would direct the flow of pulp into the desired pumpcell. Six CIP tanks in series would be provided to process flotation concentrate. Concentrate CIP tailings would be pumped to the CIL tanks. The hybrid system would allow cyanide in the CIP tailings to be used to leach gold remaining in the flotation tailings and the CIL tanks would allow additional adsorption contact to maximize soluble recovery.

17.3.7 Oxide Carbon-in-Leach and Tailings Detoxification

While the majority of the mineral resources and reserves at the Stibnite Gold Project are strongly refractory, non-refractory material is present in all three deposits, and in the Historic Tailings. To recover gold from non-refractory material in the flotation tailings, and in oxide material that would be processed during oxidation circuit scheduled maintenance periods, a CIL circuit was included in the design of the process plant.

Underflow from the neutralization thickener would be conditioned with lime in two 54 ft in diameter x 56 ft high oxide-conditioning tanks in series with a total retention time of 5 hours; slaked lime would be added to adjust the pH. Slurry from the pre-conditioning tanks would flow by gravity to the CIL tanks where it would be mixed with the tailings from the CIP circuit. Mixing the two streams would allow extended leaching of the flotation concentrate and use of the residual cyanide in the concentrate leach stream to leach the oxide material. Six 54 ft in diameter x 56 ft CIL tanks in series would provide approximately 14 hours of retention time for the combined neutralization thickener underflow and concentrate leach tailings streams.

Additional cyanide solution and compressed air could be added to the CIL tanks to facilitate gold leaching. Barren carbon from the elution circuit would be added to the tailings CIL and the loaded carbon from this scavenger stage would advance to the concentrate CIP tanks. Each CIL tank would be equipped with one operating Kemix-type carbon screen with approximately 270 ft^2 of screen area and one carbon advance pump. A monorail hoist is provided at each tank to facilitate screen changes. One standby screen is provided to allow one screen to be pulled from the process and cleaned daily. A mobile crane would be used to relocate the spare screen. CIL tailings would be screened on single-deck vibrating safety screens. Safety screen undersize would flow by gravity to the detoxification system.

Tailings from the CIL would be treated to reduce the cyanide concentration prior to discharge. In the cyanide oxidation tanks, weak acid dissociable (WAD) cyanide would be oxidized to the relatively non-toxic form of cyanate using sodium metabisulfite solution and air. Copper is normally added as a catalyst, but more than adequate copper has been added to the flotation step as an activator, so no additional copper sulfate is expected to be consumed in the detoxification process. Milk of lime would also be added to maintain a slurry pH in the range of 8.0 to 9.0. Air required for the reaction would be sparged below the tank agitators. Two 40 ft diameter x 42 ft high tanks in parallel, and in a concrete contained area, would provide a total retention time of approximately two hours for the detoxification operation. The SO_2-air method for detoxifying the tailings was shown to be effective based on the laboratory results presented in Section 13.

The detoxification circuit would reduce cyanide concentrations in the tailings slurry to less than 50 ppm WAD cyanide before being transported to the TSF. This WAD cyanide concentration target is based on guidance from the International Cyanide Management Institute (2002) as the concentration is generally accepted to protect birds, other wildlife and livestock from the adverse effects of cyanide process solutions; a lower concentration could be targeted, if required. A lower concentration may also be required to ensure high sulfide recovery in the flotation process. Since tailings reclaim water would be recycled to the mill, and the mill process includes sulfide flotation, cyanide must be reduced to low levels for efficient processing by flotation. Other processes in the TSF, including natural oxidation by UV radiation from sunlight, will continue to reduce the cyanide concentration in the tailings supernatant.

17.3.8 Carbon Handling and Refining

Loaded carbon from the CIP or CIL process would be screened and washed using a single-deck vibrating screen. Screen oversize would flow to the acid wash column and screen undersize would be returned to the CIP or CIL circuit for recovery of soluble metals. The acid wash and elution vessels would each have a 7-ton carbon capacity. Nominally 7 tons of carbon would be advanced daily allowing a loaded carbon concentration, of from 100 - 200 oz/st, depending on the feed grade. The estimated gold loadings are reasonable since the loaded carbon would be processed through the concentrate CIP process and minimal silver or copper would be recovered.

During the acid wash process, solution would be circulated and nitric acid would be added to the system to maintain the solution pH. Nitric acid is used to limit chloride ion use and build up in the system; chloride ions can cause autoclave corrosion. The nitric acid added would react with calcium carbonate that is adsorbed on the carbon and help maintain the carbon activity. The acid wash solution would be circulated to the acid wash circulation tank. When the acid wash is complete, the acid would be rinsed from the carbon with fresh water and solution would be diverted to the neutralization tank where it would be mixed with caustic to a safe pH. An exhaust fan and scrubber are provided to control hydrogen cyanide gas that is generated during the acid wash process.

Acid-washed carbon would be transferred to the elution vessel where it would be stripped of precious metals by the pressure Zadra method. Electrolyte would be pumped from the strip solution tank through heat exchangers to the elution vessel; heat would be added to the system as needed by a propane fired strip solution heater and the primary heat exchanger. From the elution vessel the pregnant electrolyte would flow through heat exchangers to the electrowinning feed tank and the three electrowinning cells, each with 2,000-amp rectifiers. Electrowinning cell tailings flow to the barren eluate tank and would be pumped back to the strip solution tank to complete the solution circuit. The precious metals and mercury recovered from the carbon would be plated onto stainless steel cathodes and recovered periodically in the refinery and pumped to a sludge filter.

The precious metal filter cake would be dried in the retort and the minor amounts of mercury that may be present in the sludge would be volatilized and recovered from the retort to ensure it does not enter the environment. Recovered mercury would be stored onsite in metal flasks prior to shipping to a safe disposal site or sold. Precious metals remaining in the dried sludge would be mixed with flux and melted in an induction furnace and poured into precious metal doré bars. Off gasses from the electrowinning cells and retort would be mixed and processed through a demister and carbon adsorption vessel. Off gasses from the induction furnace would go through a baghouse, HEPA filter, and carbon adsorption vessel. The planned retort has a 10 ft^3 capacity and is electrically heated. The refining furnace is a 175 kW electric induction furnace. A vault for secure doré storage is included in the refinery building.

Stripped and acid-washed carbon would be transferred to the kiln feed screen. Screen oversize would flow to the kiln feed bin and a screw feeder would feed the carbon to the kiln. The kiln would dry the carbon and heat it 1,290 ˚F for 10 minutes. Regenerated carbon would be returned to the CIP or CIL circuit via the carbon-sizing screen. The kiln has a design throughput of 6 tons of carbon per day. Considering the flotation reagents used, reactivation of a high percentage of the carbon is recommended.

The carbon handling and refinery area was designed as a single, 60 ft x 120 ft x 45 ft high building with two distinct areas and construction types. The ADR area is a steel and concrete building that houses the carbon regeneration kiln, the acid wash column, and all the tanks and vessels required for stripping the precious metals out of the electrolyte solution and feeding the refinery. The steel structure is supported on concrete piers and supports preformed insulated metal roof and wall panels. The refinery was designed as a masonry structure, 48 ft x 120 ft x 16 ft high. The refinery area contains the electrowinning cells, the mercury retort and the furnace.

17.3.9 Historic Tailings Reprocessing

Metallurgical testing indicates that the Historic Tailings contain significant recoverable gold; moreover, as detailed in Section 14, the average grade of the tailings is well above the economic cut-off grade. Since the tailings are within the design-footprint of the Main WRSF they have to be removed early in the mine life in order to allow the placement of waste rock from the Project.

M3 conducted a trade-off study to evaluate various methods of collecting the Historic Tailings and delivering the material to the process plant; three methods were considered: excavation, dredging, and hydraulic mining. Two transportation methods, trucking and slurry-pumping, were considered to get the material to the grinding circuit. Excavation of the material, trucking to a screening plant for re-pulping, and pumping the slurry to grinding circuit was selected as the method best suited to the material handling and environmental challenges posed by the operation.

The Historic Tailings would be mined by mechanical equipment and hauled to the re-pulping plant by trucks. Trucks would dump the material onto a grizzly screen and into the feed hopper. An apron feeder would feed a vibrating screen and screen oversize would drop to a containment bunker for periodic removal. Water would be added at the vibrating screen to facilitate the re-pulping process. Screen undersize would discharge to a sump and sump discharge would be pumped to the process plant.

Water for the Historic Tailings re-pulping system would be provided from the tailings reclaim water system. Water sprays would be added to the screen where needed to re-pulp the tailings material. An air compressor and instrument air dryer would be installed for operation and maintenance. A mobile crane would be available for maintenance of the equipment.

Based on the PFS mine schedule presented in Section 16, the ±3 million tons of Historic Tailings have to be moved within the initial 4-years of operation to avoid conflicts with the waste rock storage schedule. Based on an estimated availability of 75%, the tailings re-pulping facility was designed with an instantaneous throughput of 115 st/h.

17.3.10 Process Reagents

Reagents requiring handling, mixing, and distribution systems are summarized in Table 17.4; the table also includes estimated reagent consumption rates for full-scale plant operation, which have been estimated based on metallurgical testing results.

The dry reagents would be stored under cover, then mixed in reagent tanks and transferred to distribution tanks for process use. The reagent building would be a steel-framed structure with metal roofing; metal siding would be installed to keep reagents dry and protected from the sun. The floors would be slab-on-grade concrete with concrete containment walls to capture spills.

Table 17.4: Estimated Primary Reagent Consumption Rates

Reagent	Use in Process Plant	Yellow Pine		Hangar Flats		West End		Historic Tailings
		High Sb	Low Sb	High Sb	Low Sb	Sulfide	Oxide	Low Sb
		lb / ton	lb / ton	lb / ton	lb / ton	lb / ton	lb / ton	lb / ton
Pebble Lime (CaO)	Neutralization pH control, conditioning, leach and detox, pyrite depressant	17.6	17.2	15.0	14.4	14.4	4.0	7.0
Lead Nitrate ($Pb(NO_3)_2$)	Antimony activator	0.71	0.00	0.71	0.00	0.00	0.00	0.00
Aerophine 3418A	Antimony collector	0.03	0.00	0.04	0.00	0.00	0.00	0.00
Copper Sulfate ($CuSO_4$)	Sulfide activator and detox catalyst	0.90	0.40	0.55	0.90	0.50	0.00	0.40
Potassium Amyl Xanthate (PAX)	Sulfide collector	0.43	0.27	0.33	0.52	0.47	0.00	0.22
Methyl Isobutyl Carbinol (MIBC)	Frother	0.08	0.08	0.09	0.06	0.05	0.00	0.06
Sodium Cyanide ($NaCN$)	Gold and silver complexing agent, pyrite depressant	1.0	0.80	1.0	0.80	0.80	0.80	0.80
Flocculant	Promote settling	0.13	0.13	0.13	0.13	0.13	0.07	0.07
Activated Carbon	Recover soluble gold and silver	0.10	0.10	0.10	0.10	0.10	0.10	0.10
Sodium Metabisulfite ($Na_2S_2O_5$)	Oxidize free and WAD cyanide	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Nitric Acid (HNO_3)	Decalcify activated carbon	0.08	0.08	0.06	0.06	0.05	0.05	0.04
Caustic ($NaOH$) (sodium hydroxide)	Carbon acid wash neutralization	0.07	0.07	0.06	0.06	0.05	0.05	0.05

17.4 WATER SYSTEMS

Two types of water systems are required for the Stibnite Gold Project process plant: fresh water and process water.

Fresh water for the Project would be supplied from groundwater wells located within the Meadow Creek valley alluvial deposits. Water from the wells would be pumped to the freshwater tank, which also serves as the firewater tank; fresh water in the tank would be distributed to and used for:

- the freshwater distribution system;
- the fire water pipeline loop;
- the gland seal water tank and pumped by horizontal centrifugal pumps to be used as seal water for mechanical equipment;
- the mine water trucks to be used in road dust control; and
- the process use points (e.g. crusher dust suppression, reagent mixing, etc.).

Process water would be reclaimed from several locations and returned to the process water tank. Overflow from the neutralization thickener, gold concentrate thickener and the antimony concentrate thickener would be pumped to the process water tank. Water reclaimed from the TSF, stormwater pond, and pipeline maintenance ponds would also be returned to the process water tank.

17.5 PROCESS AIR SYSTEMS

Several of the agitated process tanks require injected air provided by blowers, including the neutralization tanks, pre-conditioning and leaching tanks, and pre-conditioning and CIL tanks for oxide. Each of these systems has a dedicated blower and installed spare to provide the necessary volume and pressure of air for the process.

Gaseous oxygen is provided to the autoclave at pressure of 570 psig to facilitate oxidation of the sulfides to liberate the precious metals. The oxygen would be supplied from a vendor-supplied oxygen plant located near the autoclave building.

17.6 TAILINGS HANDLING SYSTEM

M3 conducted a study to evaluate the methods to pump the tailings from the process plant to the TSF. The design basis involved pumping approximately 6,000 gallons per minute of tailings with 55% solids and a specific gravity of 1.53 a vertical distance of 440 feet (starter dam) to 650 feet (final dam) and a horizontal distance of approximately 19,000 feet (starter dam) to 23,000 feet (final dam). Capital and operating costs for horizontal centrifugal and positive displacement pumps were compared and the centrifugal pumps were selected on the basis of lower life-of-mine cost, primarily due to lower initial capital cost. Various pipe types and configurations were evaluated in terms of calculated pressure and friction losses. HDPE-lined carbon steel pipe was selected for the tailings pipe from the process plant to the TSF because it was the lowest cost alternative that could handle the pressure and reduce friction losses.

The tailings would be pumped using six horizontal centrifugal pumps connected in series to lift the tailings to the starter dam crest elevation of approximately 6,873 feet amsl. Six spare pumps would be installed in series to enable continued pumping if one of the pumps in the initial series should fail. The tailings would be transported in HDPE-lined carbon steel piping 24 inches in diameter in a lined trench or, when buried, in a containment sleeve. The pipeline is routed west from the thickener and crosses EFSFSR after approximately 500 feet. The pipeline routing then parallels the waste haulage road and then climbs up the slope on the northern side of the Meadow Creek valley, parallel to the surface water diversion around the WRSF. Additional information on the configuration and management of the TSF is provided in Section 18.

Supernatant from the TSF would be reclaimed and pumped via three barge-mounted vertical turbine pumps and pipeline to the process water tank located in the process plant area; the reclaim water pipeline would share the same secondary containment as the tailings pipeline. The TSF impoundment must be raised periodically to provide additional tailings capacity; the tailings pipeline would be relocated and extended to accommodate these raises. One additional pump and one spare would need to be added to the tailings pumping system as the TSF dam rises to its ultimate height of approximately 7,060 ft amsl.

The initial routing of the pipeline (and waste haulage road) transects the ultimate Hangar Flats open pit and must be moved to circumvent the pit when mining begins to encroach; Meadow Creek also has to be realigned since it transects the ultimate Hangar Flats pit. The pipeline, road, and Meadow Creek diversion would all be moved concurrently to be outside of the ultimate Hangar Flats pit.

The tailings pumping system would be housed in an 80 ft x 125 ft x 40 ft high steel-framed building supported on concrete piers with preformed insulated metal roof and wall panels. There is an overhead bridge crane for pump maintenance.

17.7 PROCESS CONTROL SYSTEMS

The Stibnite Gold Project process plant design includes an integrated process control system consisting of three tiers of control and monitoring systems. A conceptual description of the control architecture is provided below, followed by a conceptual control philosophy that depicts the level of automation and the principles that would guide decisions concerning instrumentation and control design in the next phase of this Project.

17.7.1 Process Control Architecture

Process control for the process plant would be accomplished by a multi-tiered monitoring, control, and recording system using an Ethernet backbone. The fiber optic network would be arranged in dual self-healing ring configuration for redundant peer-to-peer communications and control. The redundant fiber optic communication modules protect the integrity of the Ethernet network by maintaining network communications, even with a failure of a fiber path. The functions of the network include data collection and control on a single high-speed network, with tie-in to the plant management system. The devices on the network include servers, workstations, switches, Programmable Logic Controllers (PLCs), and Human-Machine Interfaces (HMIs).

The control system consists of three levels of control: local control, PLC control, and Process Control System (PCS) control. Local control of each piece of driven machinery is from a local hand control station, typically a station with Start and Stop pushbuttons. Field Stop pushbuttons are hard-wired directly to the motor control centers (MCC) to operate independent of the control system or selector switch position. Likewise, personnel safety features, such as conveyor pull cords, are directly connected to the motor controls. Each piece of driven machinery is equipped with a Local/Off/Remote selector switch located in the MCC. The selector switch is arranged to provide bump-less control between the local Start/Stop pushbuttons when in the Local position, and the PLC control system when in the Remote position.

PLCs control the process equipment when the local control switch is in the “Remote Mode”, and provide monitoring and control of the equipment. PLCs are accessible to both field operators and operators in the control rooms. The PLC system would monitor the status of all local controls to supervise operations and alarm the operator of any anomalies in the system’s configuration.

The PCS integrates the system components, from the device-level communications and control, to the Ethernet networks and higher-level business systems. It incorporates redundant virtual servers and operator workstations into the network to enable operators in the mill control room, crusher control room, and in other designated control stations throughout the site to monitor and control the various component processes. These workstations would be configured to access the process screens and data associated with their specific process area. Two large screen monitors installed in the mill control room provide a process overview. Access to historical process records is provided by the historian server. An engineering workstation is installed and configured with access to all process interface screens, as well as the software required to provide system configuration and maintenance.

17.7.2 Process Control Philosophy

The process plant would incorporate modern, dependable and proven instrumentation and control systems. The monitoring and control systems would support the operation of the plant under the following parameters. The plant would operate on a two 12-hour shift per day basis. Planned maintenance shutdowns would take place on a regular basis. The plant would have an overall operating availability 92%, with lower availabilities for the crusher (75%) and autoclave (85%). There are no holiday and/or other planned work stoppages during the calendar year. The maintenance of the monitoring and control systems would be performed in accordance and support of this operating and maintenance schedule.

The mill building control room would serve as the center for communications, fire systems monitoring and emergencies in general. The control room would be manned on a 24 hour-a-day basis. A base station radio would be assigned to the control room as well as an outside telephone line. The control room would also have the ability to communicate on all other site group frequencies. The control room operator would also have access to the company e-mail system.

Real time observation of strategic points along the operation would be by a TV camera system with monitors in the control room. PLC systems would be used for controlling the plant equipment. Proper graphic displays would be developed for the PLC systems. The control room would serve as the center of all control and recording of key process variables, outputs, functions and plant stoppages.

Safety systems would include, but are not limited to the following:

- The use of start-up warnings – horns, sirens or some other means – would be used throughout the property.
- Applicable interlocks would be used to protect people and equipment.
- All fire protection systems and fire detection systems would be monitored from the mill control room.
- Interlocks and/or other safety related protection would either be hard wired or in control logic depending upon which offers the greatest level of assured safety.

Real-time process control and monitoring systems that provide data to the operators would include, but are not limited to the following.

- Instrumentation on the primary crusher would provide data on power draw, weigh scale on stockpile feed conveyor, crusher discharge hopper level indicators, etc. The primary crusher would also have a tramp iron magnet and an appropriate metal detector.
- Coarse ore stockpile would have a height measuring device and the reclaim conveyor would have vendor supplied variable speed controls for each feeder.
- Each reagent system would have the ability to be batched to the necessary strength and stored until used in the plant. The delivery systems would have the ability to be measured and controlled from the plant control room.

- The grinding area instrumentation would include the SAG mill feed conveyor weight scale, water and reagent control to the SAG mill, tramp steel magnet, cyclone feed sump levels and auto water addition to the sump, pulp densities for the cyclone feed pump discharge as well as cyclone pressure, and the ball mill power draw and automatic water addition. Both grinding mills would have the vendor supplied controls, interlocks and monitors to protect the equipment.
- The flotation circuits have on-stream X-ray analyzers. The following streams would be automatically sampled and analyzed: rougher flotation concentrate, rougher scavenger flotation concentrate, rougher tailings, first cleaner scavenger flotation tailings, and 2nd cleaner flotation concentrate. Flotation sumps would have level indicators and automatic valves for water and/or reagents where applicable. Flotation cells would have the vendor supplied packages to allow level control and other needed instrumentation normally associated with their product. Thickeners would have torque indicators with adjustable height rakes and automatic valves on the thickener underflow pumps.
- The antimony filter would have all typical vendor-supplied instrumentation. A truck scale would be necessary in order to weigh antimony concentrate prior to leaving the site. An automatic wheel wash system would be needed to ensure environmental requirements are met.
- The pressure oxidation process would be controlled by a PLC housed in the mill control room. The PLC would monitor the sulfur content and slurry density from the autoclave feed tank and pressure and temperature in the autoclave. Based on those measurements, the PLC would adjust the water and oxygen addition to the autoclave and venting of CO_2 to the flash vessels.
- The oxygen plant would be vendor supplied and vendor operated. Appropriate operating characteristics and alarms would be transmitted to the mill control room through the Ethernet.
- Slurry density, temperature and pH are monitored in the CCD process to enable the PLC to control addition of wash water and lime in the neutralization and leach pre-conditioning tanks.
- Cyanide concentration would be manually monitored and adjusted.
- Reagent addition in the detoxification tanks would be automatically metered by the PLC using monitoring information from the CIP/CIL tailings.
- The ADR plant would have vendor-supplied instrumentation and controls operated by plant personnel. Key operating parameters would be monitored by the PCS in the mill control room.
- The neutralization thickener would have a torque indicator and adjustable lift rakes. All typical vendor-supplied indicators and systems are anticipated. Thickener underflow and recycle systems would have automatic valves and a flow and density meter.
- The tailings system would have horizontal centrifugal pumps and would have remote start and stop control capability from the mill control room.
- The TSF reclaim water barge would have vertical turbine pumps with remote stop and start capabilities from the mill control room. Each pump would receive a control signal from the reclaim water storage tank. The reclaim water storage tank would have a level indicator and an automatic control on the antiscalant addition line.

Process control and monitoring systems that measure, weigh, monitor, and collect samples for assaying would include the following:

- a weigh scale on the coarse ore stockpile conveyor to enable reconciliation of mine-delivered tonnage with tons crushed;
- a weigh scale on the coarse ore reclaim conveyor for the metallurgical balance;

- automatic sample cutters would be utilized to ensure samples are taken on a regular basis and the shift composite samples would serve as a basis for the plant metallurgical balance;
- appropriate flow meters, scales and control valves would be installed where deemed necessary;
- before leaving the site, antimony concentrate would be weighed and sampled for moisture and antimony content as well as gold and silver content; and
- gold doré would be weighed and sampled for precious metal and impurity contents before being shipped offsite.

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18 PROJECT INFRASTRUCTURE

18.1 INTRODUCTION

Existing infrastructure relevant to the development and operation of the Stibnite Gold Project was presented previously in Section 5. This section summarizes the results of trade-off and technical studies completed to establish appropriate infrastructure upgrades and infrastructure additions that would be required to support the mining and mineral processing facilities that were discussed in Sections 16 and 17, respectively. The Project infrastructure needs that are discussed in this section include:

- upgrades to existing roads to support safe and reliable all-season vehicle access to the site;
- off-site logistics, warehousing, metallurgical laboratory and administration facilities near Cascade;
- upgrades to the Idaho Power Company (IPCo) electrical distribution system to provide reliable, low-cost, low greenhouse gas (GHG) emissions electricity during mine operations;
- installation of an on-site power supply system to support construction activities and provide backup power during possible service interruptions to the IPCo electrical system;
- upgrades to the existing microwave communications system to provide reliable high-speed data and voice communications for construction and operations personnel;
- upgrades to the existing on-site camp to support construction and operations;
- construction of surface and contact water management infrastructure;
- fresh water, reclaim water, and potable water supply systems;
- process water treatment and management infrastructure;
- waste management infrastructure such as a tailings storage facility (TSF) and waste rock storage facility (WRSF); and
- sanitary waste management infrastructure.

Initial capital, sustaining capital, and closure costs associated with the infrastructure discussed herein are provided in Section 21.

18.2 SITE ACCESS

Vehicle access to the Project site is currently via secondary roads that intersect Highway 55 near the communities of Cascade and McCall, as previously discussed in Section 5, and as shown on Figure 18.1. In order to facilitate safe year-round access for mining operations; reduce proximity of roads to streams, creeks and rivers; and respect advice of community members, a new site access road alignment was developed that uses the existing US Forest Service road (NF-447), known locally as the 'Burntlog' Road (the "Burntlog Route"). Figure 18.1 illustrates the alignment of the Burntlog Route, which from Warm Lake follows the Warm Lake Road (FH 22) for 10.2 miles to Landmark, traverses the Burntlog Road (FS 447) for 17.3 miles before transitioning to a new road alignment for 8.4 miles that traverses through the Trapper Creek drainage basin to connect to the existing Thunder Mountain Road at the bottom of Meadow Creek Ridge. The route then follows the Thunder Mountain Road until it reaches an area where the route begins a steady decline in elevation to the Project site.

The Burntlog Route was selected over several other possible alternatives, such as the Cabin/Trout Creek and the Johnson Creek alternatives, following a comprehensive, multi-phased access road trade-off study. Provided below is a summary of the key attributes that resulted in the Burntlog Route being selected as the preferred route:

- least road length containing steep vertical grades and within avalanche and landslide potential areas;

- much less elevation loss after the first summit;
- least amount of excavation and hauling excess material to waste sites;
- least miles of newly constructed road through previously undisturbed national forest and riparian conservation areas (RCA);
- eliminates mining-related travel and transporting of materials alongside major waterways (Johnson Creek or South Fork of the Salmon River);
- minimizes the risk of hazardous material spills into major waterways (only one Johnson Creek crossing);
- least road length paralleling streams, reducing the risk of hazardous material spills and sediment load into streams;
- fewest amount of retaining walls;
- lowest cost when compared to the other short-listed alternatives; and
- it is likely to require the least amount of time to construct.

Preliminary design criteria were based on jurisdictional policies of Valley County (Valley County, 2008) and the USFS (U.S Forest Service, 2011a) except the width of the access road was decreased from 28-feet wide to 20-feet wide to reduce cost and environmental impact. This design exception would require approval from jurisdictional agencies. Key criteria for resource development road include: design speed of 20 mph, maximum 10% vertical grade, 3% cross slope, and 20-foot width.

The access road would connect to onsite roads, which include haul roads, process plant roads, and service roads associated with the tailings storage facility and other facilities on the Project site. The contemplated roads, Project facilities, and overall site layout are shown on Figure 18.2. The onsite roads would be all-weather unpaved gravel roads that would require dust suppression in the dry months, something Midas Gold does with existing roads on and near the Project site already. Haul roads would be designed to accommodate the largest truck planned, as discussed previously in Section 16.

Figure 18.1: Proposed Road and Electrical Infrastructure Upgrades

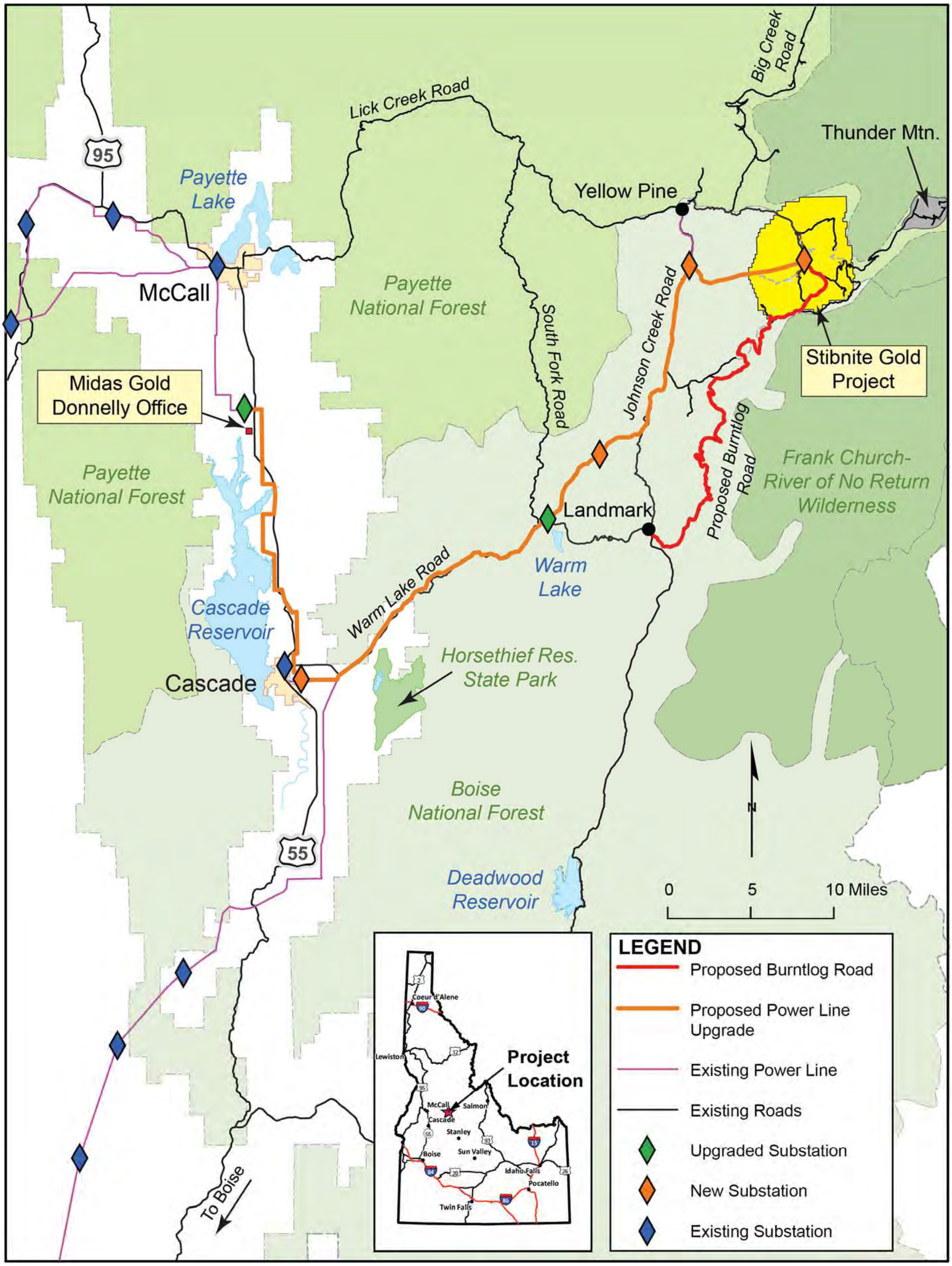
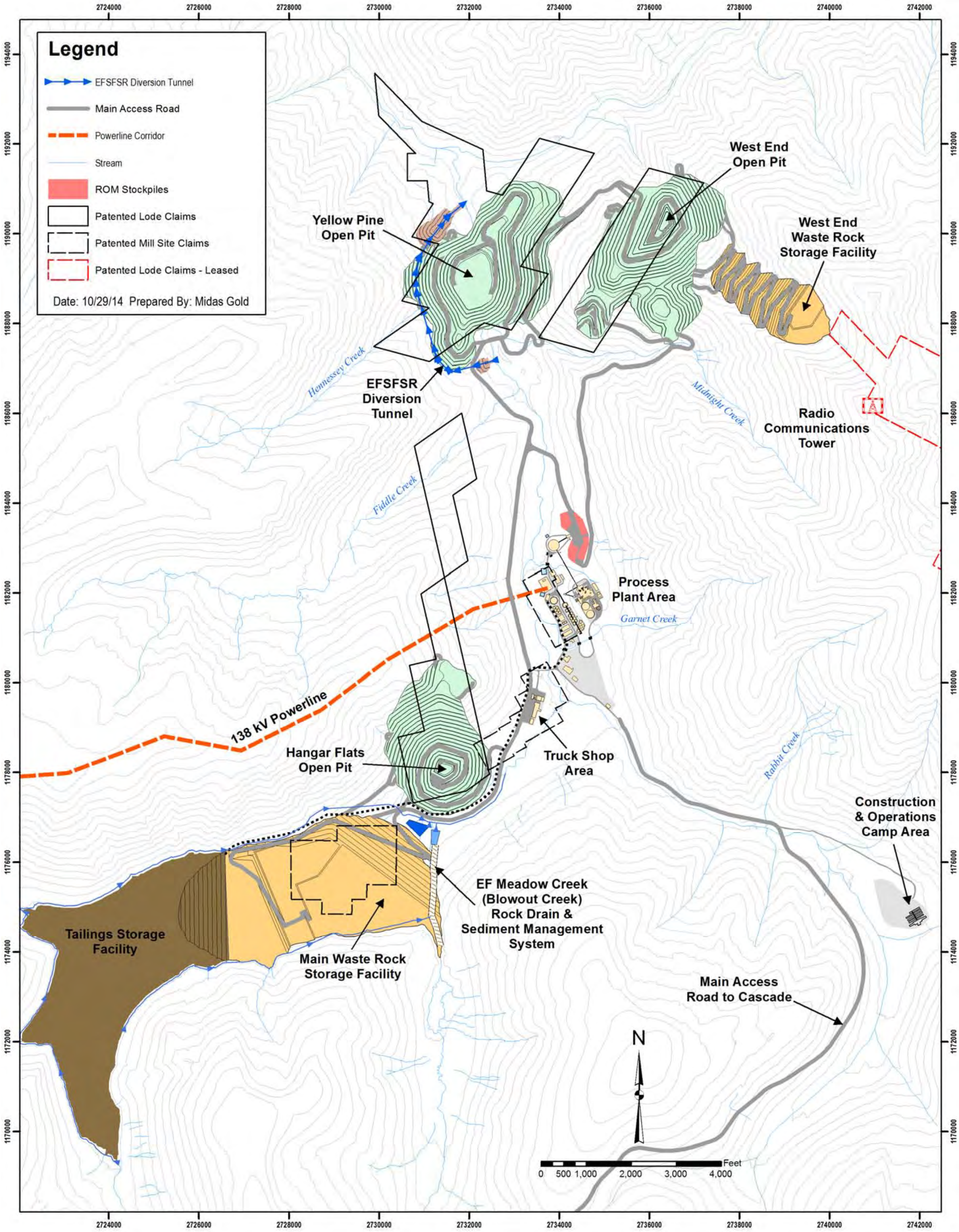


Figure 18.2: Proposed Site General Arrangement



18.3 OFF-SITE ADMINISTRATION, WAREHOUSE AND METALLURGICAL LABORATORY COMPLEX

In an effort to reduce traffic to and from the Project site and to reduce housing requirements at the site, administrative offices for the operation will be located in or near the town of Cascade (the "Cascade Complex"). The Cascade Complex would include offices for managers, safety and environmental services, human resources, purchasing, and accounting personnel. The administration building would be modular, consisting of eight 12 ft by 60 ft units. Network servers and the communications link for the mine would also be located at this complex, as well as the offsite repository for physical and electronic records for mine operations. Administration personnel in Cascade would coordinate procurement of and payment for the goods and services required at the mine site.

The Cascade Complex would also have a small warehouse to accumulate parts and supplies and a parking area for trucks to check-in and assemble prior to traveling to the Project site. Drivers would check-in at this complex and either proceed to the site, typically in a convoy, or unload at the warehouse for temporary storage and assembly of a load. A truck scale would be included to verify loads going into and out of the warehouse area, as well as a laydown area for temporary outdoor storage. A parking and assembly area for operations personnel to board buses for transportation into the mine site would also be included.

The main assay laboratory would be located at the Cascade Complex. The assay laboratory would be the primary location for sample preparation, analysis, and reporting for production, exploration, and specialty sampling for mine operations. Production samples would be delivered daily to the laboratory for processing and analysis, and the results would be transmitted electronically to mine operations and exploration personnel.

18.4 PROCESSING PLANT AND ANCILLARY INFRASTRUCTURE

The majority of the Project area is characterized by steeply sloping, mountainous terrain. Flat terrain with competent foundation conditions suitable for mine infrastructure is generally limited; these areas are typically in the valley-bottoms, near the colluvium/alluvium/bedrock contact, which is consistent with infrastructure siting by previous mine-operators. In order to establish the preferred layout for the Project process plant and related infrastructure, several potential locations were considered and evaluated against the following design constraints and considerations:

- environmental constraints (e.g. proximity to surface water and wetlands);
- regulatory constraints (e.g. land ownership or mandatory offsets from jurisdictional waters);
- topographic constraints (e.g. limited amount of flat terrain);
- geotechnical constraints (e.g. geologic hazard areas, sensitive soils, landslide areas);
- safety constraints (e.g. areas that lie in a difficult-to-mitigate avalanche paths);
- social considerations (e.g. siting a camp well-away from noise sources);
- priority Project development constraints (e.g. open pits, mine operations blast zones, tailings storage facility, waste rock storage facilities, haul roads, site access roads, high mineral resource potential);
- operability (e.g. distance from blast zones and potential mineral resources);
- efficiency considerations (e.g., establishing a logical traffic flow); and
- economic constraints (e.g. environmental mitigation, capital, operating, and closure costs).

Based on the preceding criteria, four potential process plant sites were considered: (1) the PEA plant site; (2) the former Stibnite town site; (3) the Scout Ridge site; and, (4) the former SMI mill site. Figure 18.3 shows the location of these four sites relative to other site features.

Detailed process plant and ancillary infrastructure layouts were developed for each of the four sites. The benefits and drawbacks of each location were quantitatively assessed using a detailed scoring system developed from the criteria summarized above.

Table 18.1 presents a summary of the results of the analysis.

Table 18.1: Site Layout Evaluation Results Summary

Layout Criteria	Layout Scoring Summary			
	PEA Plant Site	Old Town Site	Scout Ridge Site	SMI Mill Site
Environmental, Permitting & Social Considerations	10%	21%	21%	14%
Safety Considerations	9%	13%	12%	5%
Operational Flexibility Considerations	7%	6%	6%	5%
Capital Cost Considerations	16%	29%	29%	16%
Operating Cost Considerations	13%	19%	19%	18%
Totals	55%	88%	87%	58%

Table 18.1 indicates that the Old Town and Scout Ridge sites are strongly preferred. Given their proximity to one another, it was concluded that an optimized layout that utilizes both areas could be developed. This location provides competent foundation conditions for heavy and vibratory installations (such as the crusher, SAG mill and ball mill), a centralized layout for primary crushing and conveying coarse ore to the mills, and the best flexibility to optimize the size and location of specific plant areas with respect to each other. The area is relatively dry, is further from primary streams and the EFSFSR, the area has experienced appreciable historical disturbance, and is deemed operationally safer than other locations.

Figure 18.4 presents the primary infrastructure siting constraints in the Old Town / Scout Ridge area; Figure 18.5 presents a conceptual layout for the process plant, utilities, infrastructure, and ancillary buildings optimized on both the Old Town and Scout Ridge sites. This general arrangement has an overall north to south flow of mineralized material and concentrate. Supply truck and personnel transport traffic stay clear of the open pits and are adequately separated from both ore and waste haul truck traffic, and there is ample laydown space for storing equipment, supplies, and for erecting trucks near the mine access road. The detailed arrangement was determined on the basis of safety, environmental impacts, cost, security, noise, traffic, operational ease, and in consideration of the constraints listed above.

Figure 18.3: Potential Plant Site Locations

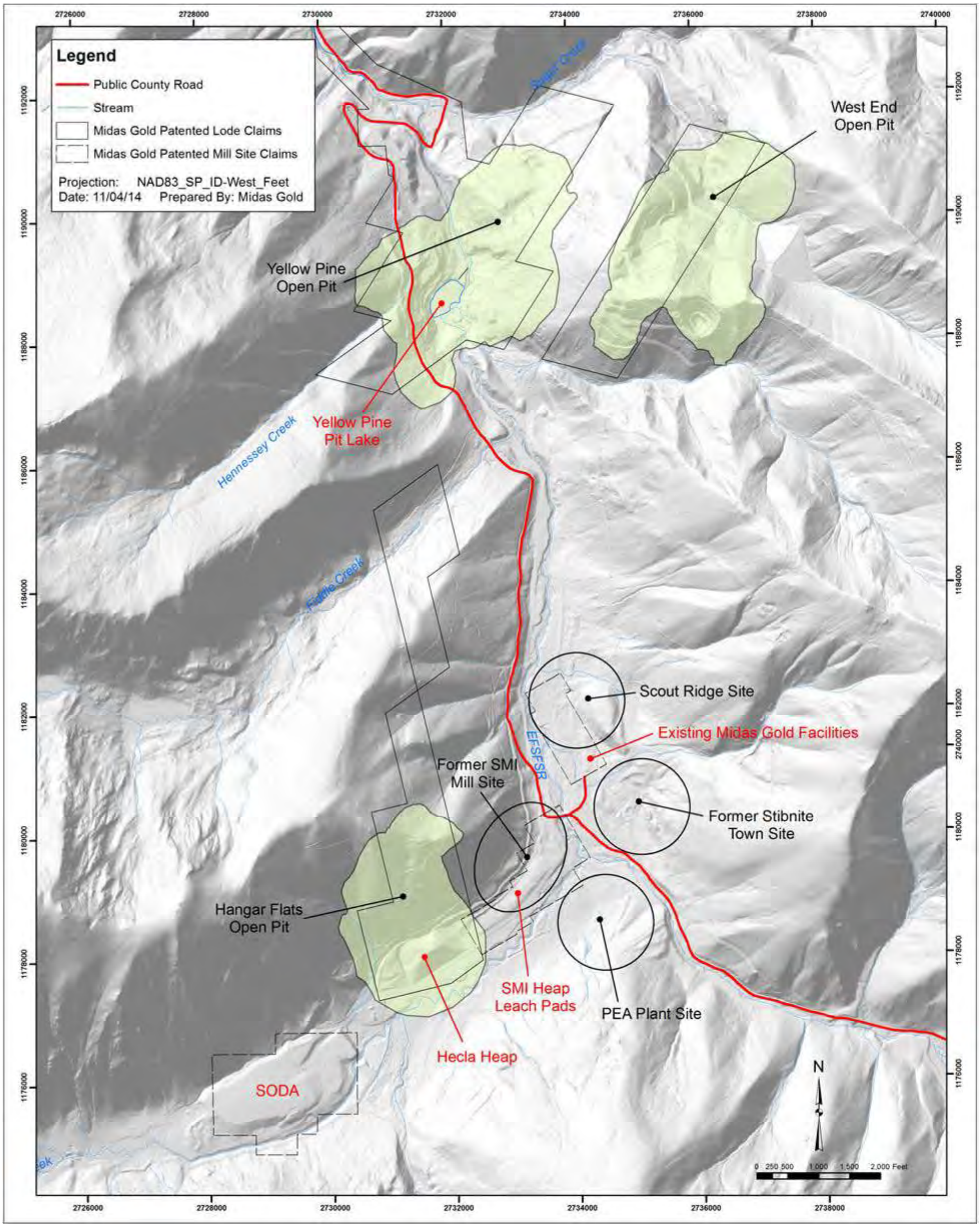


Figure 18.4: Infrastructure Siting Constraints

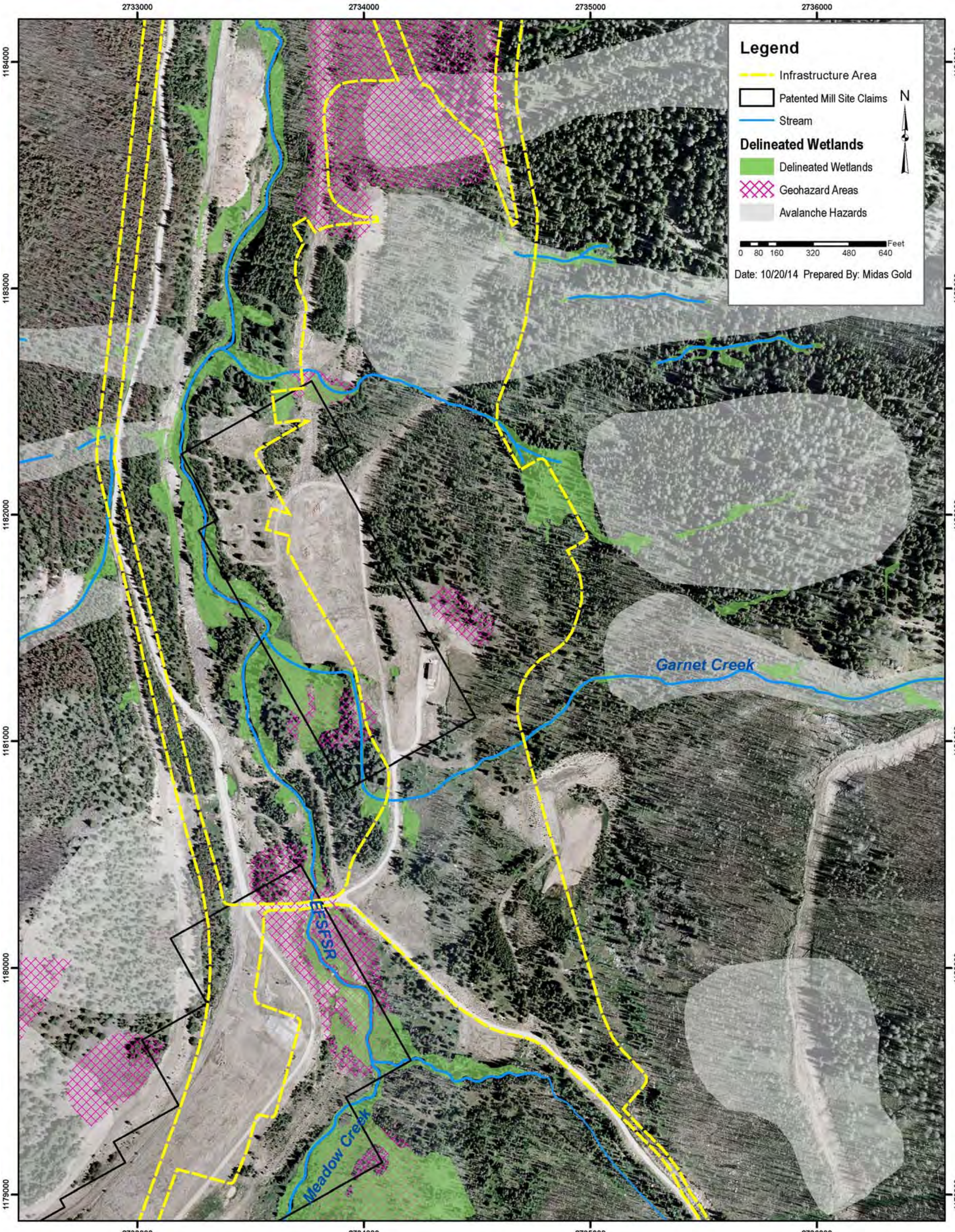
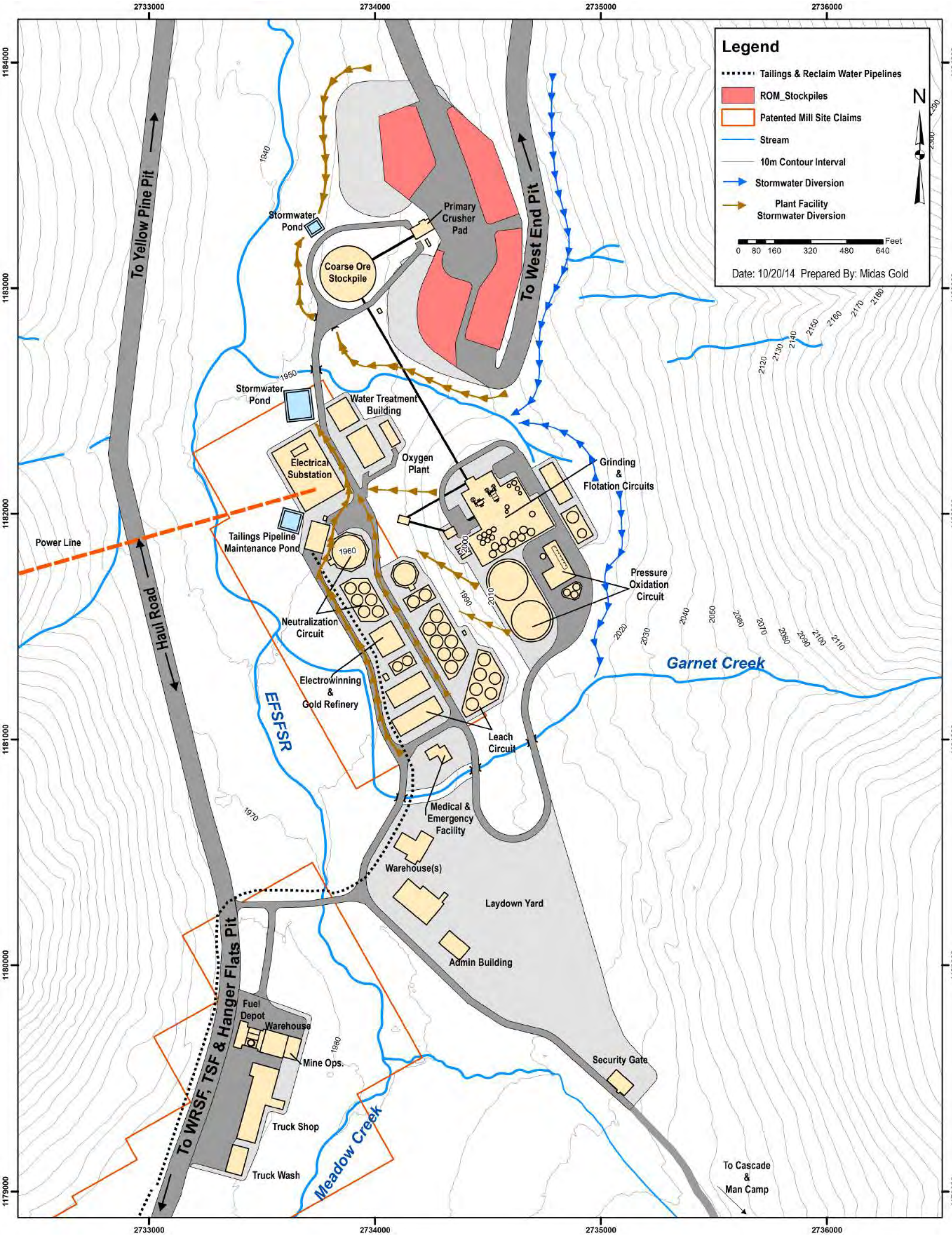


Figure 18.5: Process Plant and Ancillary Infrastructure Layout



18.5 POWER SUPPLY AND TRANSMISSION

The proposed on-site mining and mineral processing facilities are estimated to require a total instantaneous power demand of approximately 40 - 50 megawatts (MW). In order to identify the preferred power supply and distribution option, Midas Gold completed a comprehensive trade-off study that considered utility grid connection as well as both on-site and off-site self-generation scenarios including: diesel and natural gas reciprocating engines; simple cycle gas turbines, circulating fluidized bed combustion coal fired power; and renewable power generation scenarios involving photo voltaic solar, wind and hydro. In total, twelve power sources were considered and evaluated against environmental and social impacts, permitability, reliability and technical feasibility, and capital and operating costs.

Because the renewable power generation options considered would not be reliable sources of power for the Project's requirements of 24 hours per day, 365 days per year, and therefore would require significant alternative, redundant self-generation methods, grid power is deemed to have fewer environmental and social impacts as well as being the most economical alternative. A pre-existing 69 kV power-line corridor was historically permitted and constructed to the town of Stibnite, which indicates the feasibility of permitting a modern power-line to the Project site. Future studies will consider the expanded utilization of renewable energy, such as solar power which is currently being used for field operations, for areas like camp and offices, increasing the proportion of renewable energy utilized by the site.

The closest grid power-line to the Project site is a 12.5 kV distribution line supplying power to the nearby town of Yellow Pine, and the closest transmission line is a 69 kV line that provides power to Cascade and Warm Lake, Idaho. Since both power-lines are inadequate to carry the expected Project loads, the existing system would need to be upgraded to provide the additional service capability required.

The upgrades required to integrate the large load into the IPCo network include: an increased 230/138 kV transformer capacity; approximately 42 miles of 69 kV lines upgraded to 138 kV; approximately 21.5 miles of 12.5 kV line to upgraded to 138 kV line; and approximately 8 miles of new 138 kV line. Additionally, new or upgraded 138 kV substations at Lake Fork, Cascade, Warm Lake, and Yellow Pine, as well as measures to strengthen the voltages on the IPCo system are required. In addition, IPCo would need to re-supply small consumers between Warm Lake and Yellow Pine via a replacement 12.5 kV line as shown on Figure 18.1.

The 138 kV line would be routed to the Project site's main substation (the "Main Substation") where transformers would step the voltage down to the distribution voltage of 24.9 kV. The main substations would have redundant dual 138 to 24.9 kV transformers to prevent loss of power due to failure. Current Project design entails oxygen being supplied by a third party through a Sale-of-Gas (SOG) contract; therefore, a metered 24.9 kV line would be provided for the operator of the oxygen plant.

Power distribution from the Main Substation to various Project facilities would be at 24.9 kV. Main power corridors for the process plant power distribution (primary crusher, oxygen plant, truck shop, autoclave, permanent camp) would be overhead. Power within the process plant area will be underground in duct banks.

During construction, power supply would be provided by three 1,000 kW propane or diesel generators operating at 4,160 volts; the generators will also be reused for backup/emergency power during the operations phase of the Project. After primary power is provided via the 138 kV power-line, two of the generators would be relocated to the main substation for emergency power and the third would be relocated to the on-site permanent camp (as discussed below).

18.6 COMMUNICATION SYSTEM

Midas Gold's existing microwave relay (detailed in Section 5) was designed and constructed to be scalable to accommodate potential future increases in communications requirements. The system as it is currently setup provides up to 200 Mbps of bandwidth that is adequate to meet the needs of an approximately 120-person camp. Upgrading the current system to allow increases in communications capacity is a straightforward process of:

1) anchoring the existing tower pad;
2) adding an additional 20 ft section to the existing tower;
3) upgrading the antenna size to an eight foot dish or potentially adding a second antenna; and,
4) installing new high frequency (and likely FCC licensed) radios capable of increasing bandwidth to approximately 1,000 Mbps.

Updating the existing microwave relay system would provide sufficient communication capacity to service mine operations as well as the estimated 1,000-person workforce required during the Project construction period. The location of the existing tower is shown on Figure 18.2.

18.7 ON-SITE CONSTRUCTION AND OPERATIONS CAMP

Since the Project is located in a relatively remote area of Idaho, consideration was given to sourcing personnel required to construct and operate the Project as well as the housing requirements to do so.

Staffing the entire construction and operations workforce from the major nearby population centers in Valley County (population ±9,500) of McCall (population ±3,000) and Cascade (population ±1,000) has the potential to eliminate the need for an on-site residential facility (camp). While this scenario would offer significant financial advantages to the Project, the substantial one-way commute times (by road) of a minimum of 2½ hours from McCall and 2 hours from Cascade in summer conditions with an additional ½ hour in winter conditions, as well as related increased traffic, resulted in a decision that on-site housing would be required. As noted above, where possible certain functions would be located in Cascade.

Several potential camp locations were evaluated on the basis of environmental impacts, safety, cost, security, noise, traffic, quality-of-life, and operational ease. The location selected for the camp is approximately 1½ miles south-east of the confluence of the EFSFSR just off the existing Thunder Mountain Road; the location is quiet, yet located close enough to the site to yield minimal commute times, which should assist in attracting skilled operators to this remote location. For convenience, the construction camp will also be located near the operations camp area. The following sections describe how the construction and operations camps could be developed.

18.7.1 Construction Camp

Midas Gold has been conducting exploration activities at the proposed Project location since 2009 and, as a result, has facilities on-site capable of housing workers. The current on-site camp facilities are located near the proposed future plant site location and include:

- a 60-person (maximum) housing facility;
- a kitchen/dining building capable of serving 125 workers per 12 hour shift;
- a public drinking water system capable of treating 6,250 gal/day average, and a peak of 12,500 gal/hour.
- a Membrane Bioreactor (MBR) sewage treatment facility with a nominal treatment rate of approximately 9,000 gal/day average, and a peak treatment rate of 18,000 gal/day; and
- power provided by a 455 kW C-15 Caterpillar diesel generator.

To manage the estimated 1,000-person construction workforce, the existing exploration camp would be relocated and expanded appropriately. The camp would be developed based on the following assumptions:

- each room will have two beds and locking storage facilities for 4 workers who “share” the room on alternating shifts (day and night) and work cycles;

- there will be one bathroom for every 2 bedrooms;
- supervisors will have a dedicated room that is not shared with rotating shift personnel;
- the Owner's Team will be housed separately in the village of Yellow Pine.

18.7.2 Operations Camp

The operations camp would be developed by upgrading the construction camp. Approximately 517 employees are needed for the operation based on the overtime scheme associated with a modified "14 on, 14 off" work cycle. The bed count associated with this position assessment is approximately 250. As a result, the camp is designed to be a 300-person site residential facility, leaving approximately 50 beds for visitors and/or temporary workers of various types.

The distances from Cascade and McCall are too far for regular commuting from town to the Project site. Charter buses will be used to transport employees to and from the Cascade administration office/staging area and the Project site at the beginning and end of their work cycles, taking approximately two hours under good weather conditions. A charter bus company will operate a small fleet of 50- to 60-person buses, working on a schedule of staggered work cycles that will minimize the number of buses needed to handle the work cycle rotations.

Onsite transport of employees from the operations camp to the mine and plant work facilities would be accomplished by a small fleet of converted school buses and 14-person vans; the distance to transport employees from the operations camp to the various work facilities ranges from one to two miles. Operations personnel would double as bus and van operators. The onsite fleet would be winterized to handle snow conditions between the operation camp and the work areas.

18.8 SANITARY WASTE MANAGEMENT

Sanitary waste management would be handled by packaged sewage treatment facilities. The plant area and permanent camp would each have separate sewage treatment plants connected to leach fields for the treated water. The designs of the packaged sewage treatment plants are based on the calculated peak flow to the system. The leach fields would be designed in accordance with Valley County and State of Idaho standards using test work to establish the infiltration rates at the site of the leach field. Portable chemical toilets would be located at the mine open pits and other remote locations. The portable toilets would be serviced by a local vendor.

18.9 WATER MANAGEMENT INFRASTRUCTURE

Water management infrastructure is needed at the site to divert surface water around mine features and infrastructure or to control water that comes in contact with these features. Surface water that comes in contact with materials that have the potential to introduce mining and process-related contaminants (contact water) is kept separate from surface water that originates from undisturbed, uncontaminated ground (non-contact water). This is accomplished by diverting clean water around mine facilities and collecting and treating or reusing contact water.

18.9.1 Non-Contact Water Management

Surface water management activities include diversion of non-contact water originating offsite around mining operations, management of sediment from erosion occurring in the East Fork of Meadow Creek, and collection and treatment or reuse of contact water.

18.9.1.1 Surface Water Diversions

Surface water diversions are required to prevent offsite clean water from commingling with contact water, and to prevent the accumulation of excess water in the TSF. The principal surface water diversion routes Meadow Creek around the TSF and WRSF. Additional, smaller-scale diversions are provided to intercept hill-slope runoff around the perimeter of the TSF, WRSF, Historic Tailings reprocessing operation, open pits, and process plant area. Lower Meadow Creek would be diverted around the Hangar Flats pit prior to mining Hangar Flats below the creek level. Surface diversion channels are sized to convey the runoff from the 100-year, 24-hour storm event, determined using rainfall-runoff modeling. Diversion channels are either constructed in rock cut (on steep hillsides), or lined with rock riprap and geo-synthetic clay liner (GCL) to prevent erosion and minimize seepage (within alluvium/colluvium or fill). To maximize efficiency of the diversions while controlling capital costs, the TSF and WRSF diversions are phased to coincide with the various construction phases of the two facilities. TSF and WRSF diversions plans are shown on Drawing 18.1. Typical cross-sections are shown on Drawing 18.2.

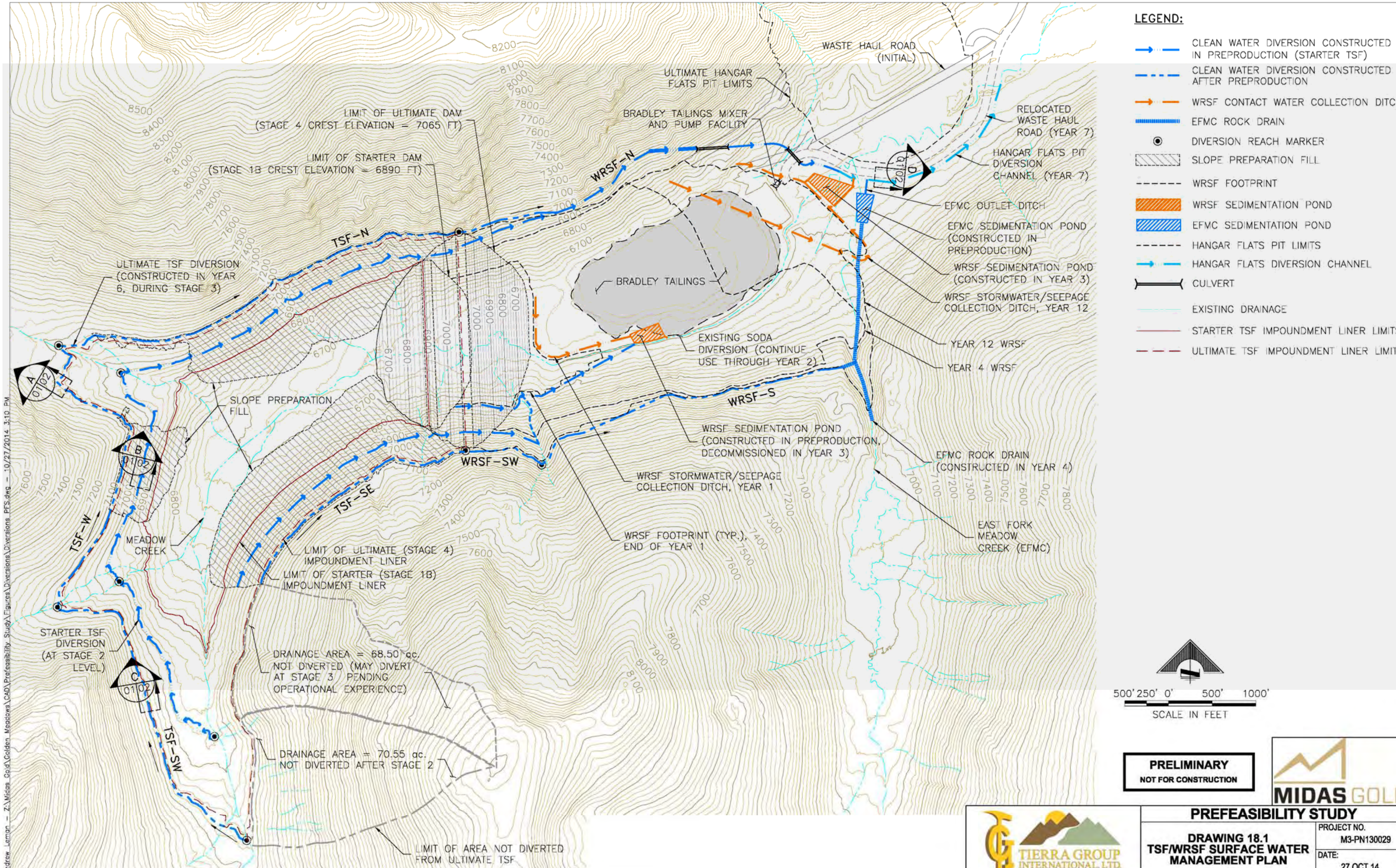
18.9.1.2 Diversion of the EFSFSR

The EFSFSR was originally diverted from its natural alignment by the Bradley Mining Company, when it began open pit mining operations in the EFSFSR valley bottom in 1938. Surface channels were constructed initially, and later the Bailey tunnel diverted the river through bedrock around the east side of the historic Yellow Pine open pit. Currently the EFSFSR flows over a steep waterfall, which is a fish migration barrier, and into the historic Yellow Pine open pit, forming a pit lake and then exiting northward to its confluence with Sugar Creek. With a future goal of re-establishment of a more natural gradient suitable for fish passage in the EFSFSR flowing in the area of the Yellow Pine open pit, the pit lake must be dewatered and the EFSFSR temporarily diverted around the pit during future mining operations. The orientation of the Yellow Pine open pit relative to the surrounding steep terrain makes a surface diversion impractical; hence, the EFSFSR will be diverted around the open pit in a tunnel founded in rock.

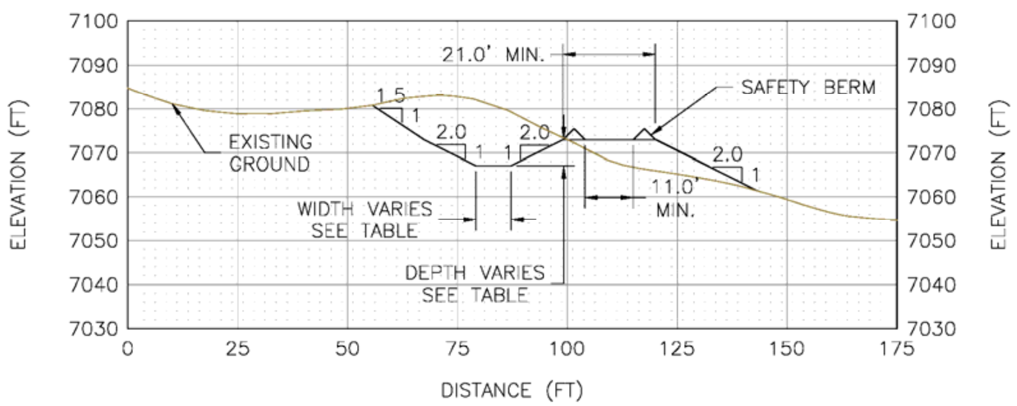
A tunnel alignment around the west side of the Yellow Pine open pit was preferred over an alignment to the east as it would preclude potential surface water impacts to Sugar Creek, which is characterized as sensitive salmon spawning habitat. The 0.8-mile long EFSFSR diversion tunnel would be 15 x 15 feet and feature a low-flow channel excavated in the tunnel floor, as well as LED lighting, to provide for and encourage passage of migrating salmon, steelhead, and trout to the headwaters of the EFSFSR for the first time since 1938, when mining commenced in the Yellow Pine open pit. Approach channels would be rock-lined, with fish resting features. The tunnel hydraulic capacity exceeds the 500-year flood event (determined from analysis of the Stibnite USGS gauge), while the low-flow channel is sized to allow maintenance access outside of the spring runoff period. Drawing 18.3 and Drawing 18.4 present the tunnel design.

18.9.1.3 East Fork Meadow Creek Sediment Control

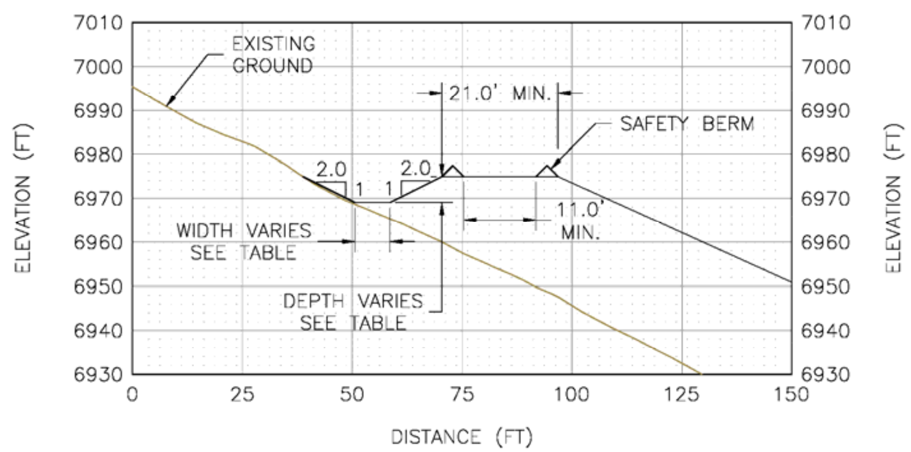
The East Fork of Meadow Creek (EFMC), which is commonly known as “Blowout Creek”, introduces a significant sediment load to the EFSFSR due to ongoing erosion within the gully and alluvial fan created by a historical dam failure in the upper EFMC watershed. The sediment degrades the quality of the gravels for Salmon redds in Meadow Creek. A sedimentation basin would be excavated in the EFMC alluvial fan and this will serve to re-establish the sediment collection function of the present Yellow Pine pit lake as it is dewatered during the early phases of the Project operations. Later, the EFMC gully and alluvial fan will be in-filled and covered with waste rock as part of WRSF construction, thus capping the source of sediment and preventing subsequent erosion and sediment deposition in the river. A rock drain will convey the EFMC under the WRSF. At the time of this writing, sediment transport measurements are ongoing within Meadow Creek, EFSFSR, and EFMC to better identify sediment sources and quantify sediment loading in the site waterways. Results of the field measurements will be incorporated into design of EFMC sediment control measures when available.



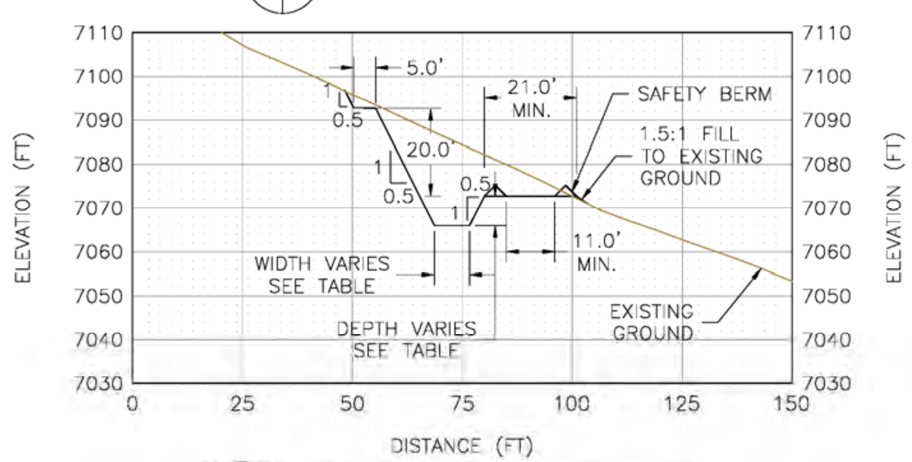
Drawing 18.1: TSF and WRSF Surface Water Management Plan



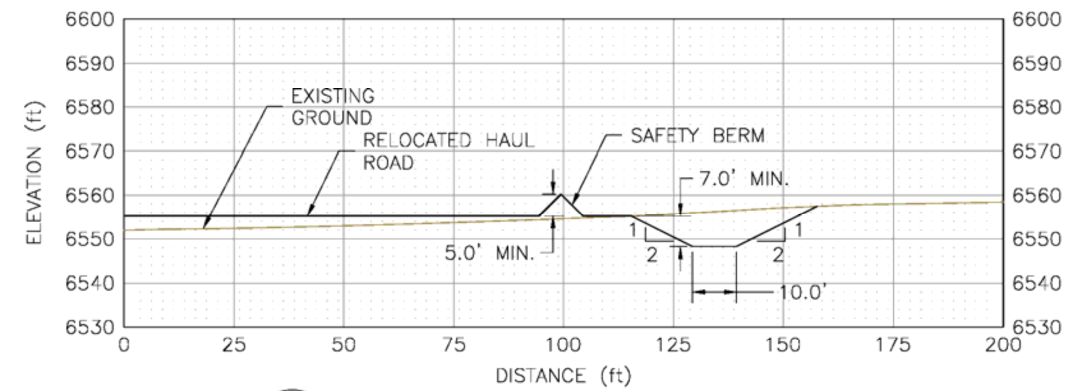
A 01|02 DIVERSION SECTION – ALLUVIUM/COLLUVIUM
SCALE AS SHOWN



B 01|02 DIVERSION SECTION – FILL
SCALE AS SHOWN



C 01|02 DIVERSION SECTION – ROCK CUT
SCALE AS SHOWN



D 01|02 HANGAR FLATS PIT DIVERSION SECTION
SCALE AS SHOWN

CHANNEL DIMENSIONS				
SEGMENT	SUBGRADE MATERIAL	SIDE SLOPE, Z (H:V)	MIN. DEPTH, D (FT)	BOTTOM WIDTH, W (FT)
TSF–SW	ALLUVIUM/COLLUVIUM	2.0:1	5.2	8.0
	ROCK	0.5:1	6.7	8.0
TSF–W	ALLUVIUM/COLLUVIUM	2.0:1	5.9	8.0
	ROCK	0.5:1	7.8	8.0
	FILL	2.0:1	5.9	8.0
TSF–N	ALLUVIUM/COLLUVIUM	2.0:1	7.1	8.0
	ROCK	0.5:1	9.8	8.0
	FILL	2.0:1	7.1	8.0
WRSF–N	ROCK	0.5:1	9.0	10.0

PRELIMINARY
NOT FOR CONSTRUCTION

MIDAS GOLD

TIERRA GROUP INTERNATIONAL, LTD.

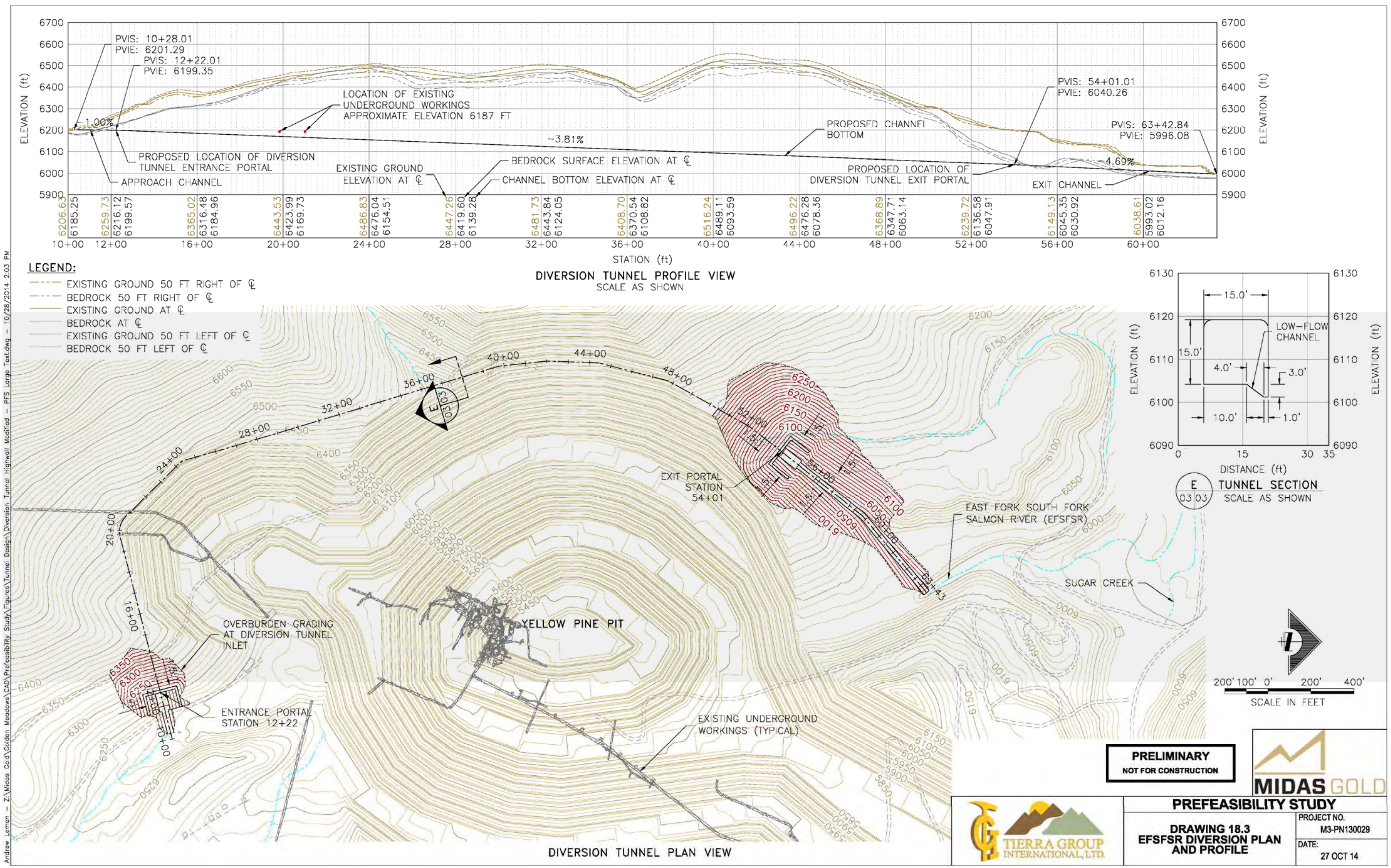
PREFEASIBILITY STUDY

DRAWING 18.2
TSF/WRSF WATER DIVERSION TYPICAL SECTIONS

PROJECT NO. M3-PN130029

DATE: 27 OCT 14

Drawing 18.2: TSF and WRSF Surface Water Diversion Details



Drawing 18.3: EFSFSR Diversion Plan, Profile, and Section

18.9.2 Contact Water Management

Contact water is surface water that has come into contact with the mine pits, ore stockpiles, spent leached ore (SODA and Hecla heap, once cover material is stripped), Historic Tailings, waste rock, or any other mining-related surface. Contact water may require active or passive treatment during construction, operations, or closure prior to discharge to the environment. Process water (including reclaim water pumped from the TSF) is addressed separately from contact water. Precipitation or dust control water falling on the listed surfaces will be collected, contained and segregated from surface waters that are not in contact with mine facilities.

Contact water from the plant site, ore stockpiles, WRSF, SODA/Historic Tailings reprocessing operation, and Hecla heap would be collected and contained in ponds or sumps sized appropriately for their respective catchment area. Water would be retained in these ponds to settle sediments, then pumped to the tailings impoundment or discharged after testing has confirmed that discharge limits are met.

Contact water originating in the pits (including surface runoff, snowmelt, and groundwater seepage) would be collected in sumps within the pits, and pumped out as needed for use in dust suppression in the pits and process makeup. Surplus contact water collected in the pits would be evaporated, treated for discharge, or pumped to the TSF for future use as reclaim/process makeup.

Runoff from roads with the potential to be in contact with process reagents would be collected. Storm water from other roads outside of the plant site, stockpiles, and WRSF area would be treated locally with small-scale sediment control BMPs to remove sediment prior to discharge. Vehicles leaving the mine site via the mine access road would pass through a wheel wash station and the wash water would be collected in a sump and treated for discharge, or pumped to the TSF for reuse.

18.10 WATER SUPPLY

Water supply for the mine, process plant, and permanent camp would be provided by three types of water systems: freshwater, reclaim water, and potable water. Freshwater for the process would be supplied from groundwater resources by a water supply well field and Hangar Flats dewatering well. Reclaim water would be reused and pumped from the supernatant water pond in the TSF. Potable water for the office and other mine facilities would be supplied by the process fresh water supply well field. A separate water supply well would be developed nearby for potable water supply to the camp. Potable water would be filtered and chlorinated before use.

18.10.1.1 Water Supply Well Field

Freshwater for process needs would be supplied by a water-supply well field located in the Meadow Creek valley, upstream from its confluence with the EFSFSR. Groundwater pumped from these wells would be collected in an equalization tank and pumped to the freshwater/firewater head tank located on higher ground east of the plant site. Partly as a safety measure, freshwater for process needs would be drawn by gravity from the freshwater tank from an elevated nozzle to allow the water in the bottom of the tank to remain available for fire suppression use, thereby ensuring an adequate water supply and pressure from gravity for fire suppression at all times, even when there is no power.

A portion of this flow would be diverted to the potable water tank equipped with a filter and chlorination system to inhibit bacteria in the potable water system.

Water for the permanent camp would be obtained from a separate water supply well located in the EFSFSR valley to its southwest. This water will be filtered and chlorinated for cleaning, cooking, showering, and consumptive use in the permanent camp.

18.10.1.2 Reclaim Water System

Process water pumped from the TSF supernatant water pond would be reused as process water. Water reclaimed from the TSF would be pumped to the reclaim water tank at the plant site. From the TSF to the plant site, the reclaim water pipeline would share a secondary containment trench with the tailings pipeline. At the plant site, the reclaim line would diverge and is located in its own containment trench. Water from the reclaim water tank would be distributed to the various points of use in the process where it is needed.

18.11 PROCESS WATER TREATMENT AND MANAGEMENT INFRASTRUCTURE

The mining operation would operate on a negative water balance during the initial phases of operation, using freshwater makeup from the water supply well field. Development of the Hangar Flats open pit in the alluvium of the Meadow Creek valley would require dewatering to limit water infiltration to the pit and maintain stability of the pit slopes. The current calculation of the site-wide water balance indicates a surplus and water would need to be evaporated or treated and discharged. A water treatment plant may be installed to treat this water to meet discharge standards. Water treatment standards are expected to include metal and particulate concentration standards, and temperature control. The design of the treatment system would be based on the characteristics of the water to be treated. In general, treatment and release of pumped groundwater (if treatment is required) would be prioritized ahead of treatment and release of contact water. Process water would be reused in the process plant and not discharged. The cost for a typical water treatment system of the type envisaged for this site is included in the cost estimate as sustaining capital.

Enhanced evaporation, using snowmaker-style misters, may be used to supplement the treatment system, in particular to prevent surplus process water accumulation in the TSF. Treatment and enhanced evaporation differ in their relative effectiveness, efficiency, usefulness in cold/wet conditions, and applicability to variable inflow water quality. Midas Gold will work with regulatory authorities and Project stakeholders to develop the most appropriate water management solution to maintain the stream flow regime and water quality of the EFSFSR.

18.12 MINE WASTE MANAGEMENT

Mine waste requiring on-site management includes waste rock from the three open pits; flotation and POX tailings from ore processing; and existing historic mine waste (spent heap leach ore from SODA and the Hecla heap) exposed during construction and mining. The existing Historic Tailings would be reprocessed, removing the majority of metals and sulfides of potential concern through flotation, and commingled with the rest of the tailings. Section 20 discusses the waste rock characterization, geochemistry, and implications for waste management. Also included in Section 20 is a characterization of the tailings and the historic spent ore from the SODA area.

Based on previous siting optimization and tradeoff studies, volume estimates from the current mine plan, and the waste characterization described in Section 20, a single TSF would be constructed to retain all tailings from the processing of the various ore types. This option is optimal to reduce Project footprint, provide for a single containment facility for monitoring and closure, and allow for the utilization of waste rock to buttress the TSF. Waste rock would be deposited in a WRSF adjacent to and abutting against the TSF, used as rockfill in TSF construction, placed as backfill within mined-out areas of the open pits to facilitate closure and reclamation, and above the current West End WRSF. Spent ore and waste rock from previous on-site operations would be used as a construction material in the TSF. Based on preliminary geochemical testing results, the construction use of spent ore material from the SODA area would be limited to applications where the material would either remain within containment or in a situation where the exposure of the material to air and water is limited. For example, placement of the spent ore below a synthetic liner but above the water table would reduce the potential for further oxidation and mobilization of constituents from this material. Reuse of this already mined material would reduce the quantities of materials required to be mined and crushed in order to construct these facilities, thereby reducing the environmental impact that would otherwise be required for mining and crushing.

18.12.1 Tailings Storage Facility

As currently envisioned, the Project would produce approximately 98 million short tons of tailings over a 12-year mine life. As the tailings would contain trace amounts of cyanide, and metals (particularly arsenic and antimony from minor amounts of sulfides not recovered in flotation), a fully-lined containment facility, utilizing a geo-synthetic liner, is proposed in order to isolate the tailings and process water within the impoundment. The tailings facility as contemplated consists of a rockfill dam, a fully-lined impoundment, and appurtenant water management features. The WRSF is located immediately downstream of and abutting against the TSF dam, and would act as a 0.6 mile-thick buttress, substantially enhancing dam stability. Design criteria were established based on the facility size and risk using applicable regulations and industry best practice for the TSF on a standalone basis; the addition of the downstream waste rock buttress substantially increases the safety factor for the design. Table 18.2 lists the design criteria for the TSF.

Table 18.2 TSF Design Criteria

	Parameter	Criteria	Comment
Solution and Water Management	Inflow Design Flood (IDF) – Impoundment	Probable Maximum Flood (PMF)	Facility will provide storage capacity above the normal operating pool to store the IDF, assuming diversions fail at the onset of the storm. No operational spillway is included.
	IDF - Diversions	1% probability (100-year, 24-hour event)	Diversions will pass peak flow from IDF without damage
	Freeboard – Impoundment	4 feet	Dry freeboard above stored IDF
	Freeboard – Diversions	1 foot	
Geotechnical Stability	Static Factor of Safety (FOS)	1.5	
	Pseudo-static (Earthquake) FOS	1.0	
	Design Earthquake	475-year (during operations); Maximum Credible Earthquake (post-closure)	

The TSF dam would be constructed of compacted mine waste rock, with a geo-synthetic liner on the upstream face (identical to the impoundment liner discussed below). Rockfill is placed in zones of successively more stringent lift height and compaction criteria approaching the liner, with the final buffer zone (directly under the liner system) consisting of gravel and smaller-sized material derived from SODA and screened historical waste dump material, overburden or valley alluvium. Construction from rockfill is inherently lower risk than construction from tailings material, as rockfill will not fluidize if saturated. TSF staging was determined from planned production rates coupled with the TSF water balance and tailings density estimates derived from consolidation testing/modeling. The impoundment would be fully lined to the elevation of the first stage during preproduction; however, the starter dam would initially be constructed at a lower elevation to balance rockfill needs with the available waste from the Yellow Pine open pit. The starter dam would then be raised during Year 1 of production to match the lined elevation of the rest of the facility. Four total stages are envisioned, with a facility expansion planned every 3 years during operations. Drawing 18.5 shows the proposed TSF dam stages and zones.

The TSF impoundment (including the upstream dam face) would be lined with geo-synthetic materials to prevent seepage of process water or transport of tailings out of the facility. The primary liner will consist of 60-mil (1.5 mm) linear low-density polyethylene (LLDPE), which features superior puncture resistance and elongation characteristics among typical TSF lining alternatives. A GCL will be placed as a secondary liner, providing a self-sealing barrier to leakage should the primary liner be torn or punctured. Where suitable soil exists (typically in valley bottoms) it would be scarified and re-compacted to prepare the liner subgrade. Steep, rocky hillsides (approximately 1/3 of the TSF footprint) would be covered with slope preparation fill to cover rock outcrops and flatten slopes sufficiently to allow liner placement. Slope preparation fill would consist of alluvium, colluvium, previously-mine rock, or rock borrowed from within the limits of the open pits. SODA material, screened site soil, or screened mine waste would be placed as a buffer zone as needed to cover coarse or rocky sections of subgrade or slope preparation fill. Slope preparation fill areas within the impoundment are designed to be stable under the same criteria as the TSF dam (static FOS 1.5, earthquake FOS 1.0).

Tailings would be deposited in the TSF from a series of drop-pipes (spigots) originating from the tailings distribution header along the facility perimeter bench. Sub-aerial tailings deposition would promote drying and consolidation of the tailings. Rotating active deposition points would allow additional drying, and sequencing of deposition would allow gradual development of a tailings beach that slopes generally from west to east within the facility, mimicking the pre-Project valley drainage and simplifying facility closure. Development of a tailings beach would also provide a measure of protection against floating ice from damaging the liner system. Drawing 18.6 and Drawing 18.7 show the TSF deposition plan for selected stages. TSF water management facilities include diversions, drainage systems, the reclaim system, and evaporators. Surface water diversion channels would serve to temporarily divert portions of the Meadow Creek within the TSF footprint and its impacted tributaries around the TSF and WRSF, while underdrains constructed in valley bottoms would collect springs and seeps and prevent accumulation of water under the liner system. A gravel over-liner drain system would collect tailings consolidation water, and route it to a sump from which it would be pumped back to the supernatant pool. Water would be reclaimed from the facility pool via barge-mounted pumps, and returned to the process plant via a pipeline. Snowmaker-type evaporators may be installed at the TSF to dispose of excess water introduced to the system when mining of Hangar Flats begins. Drawing 18.1 shows the contemplated water management plan for the WRSF and TSF. Table 18.3 summarizes the TSF design.

Table 18.3 Summary of TSF Design

Design Aspect	Description
Subgrade	Re-worked and compacted in situ materials, or minimum 12 inches of buffer/liner bedding fill.
Secondary Liner	Geo-synthetic clay liner.
Primary Liner	60-mil Single-sided textured LLDPE Geo-membrane liner.
Leak Detection	None. Underdrains may provide incidental detection and collection.
Overdrain	Discontinuous gravel drain on valley floor; geo-synthetic strip drains as needed on hillsides.
Underdrains	Geotextile-wrapped gravel trenches, with perforated HDPE pipe as needed.
Deposition Strategy	Sub-aerial; depositing from west side of impoundment and dam with pool on east side near, but not normally in contact with, dam.
Reclaim	Pumped from barge (vertical turbine pumps).

18.12.2 Tailings and Reclaim Water Pipeline Corridor

Tailings for the Project would be pumped from the Tailings Neutralization Thickener to the crest of the starter dam and then around the perimeter of the TSF. The tailings pipeline and pumping system would require sufficient head to deliver tailings to the back of the TSF. The tailings system must have enough flexibility to increase in total dynamic head as the tailings dam is contemplated to grow in height over the 12-year Life-of-Mine. In approximately Year 5, the tailings pipeline would be rerouted to the southeast to accommodate the growth of the Hanger Flats open pit.

Horizontal centrifugal pumps that increase in number as the dam height increases would be used to pump the tailings from the thickener to the TSF. The initial requirement includes four operating pumps and four standby pumps. The ultimate configuration would include six operating pumps and six standbys.

The tailings pipeline would be HDPE-lined, 24-inch carbon steel pipe. The pipeline would be installed in a geo-synthetic lined containment trench to avoid potential release of any spillage or leakage to the environment. The trench would have emergency containment ponds in low points to collect any leakage or storm water that falls within the trench. The tailings line would be installed on concrete "sleepers" to keep it off the ground. An 18-inch HDPE reclaim water line would be collocated in the trench to provide secondary containment of water being reclaimed from the TSF. The slurry line from the Historic Tailings recovery operation would also share this trench until it is no longer required.

The proposed routing of the tailings pipeline is designed to follow the waste haul road on the north side of the Meadow Creek valley (Figure 18.2). The pumping station would be on the west side of the plant area. The tailings line would be routed across the EFSFSR on a bridge in a double-contained pipe, then generally follow the haul road toward the dam. After passing the vicinity of the future Hangar Flats pit, the pipeline corridor would be installed in a trench that climbs the slope on the north side of the valley. The pipeline corridor would be accompanied by a roadway to enable monitoring and servicing the pipeline and trench. The pipeline would be installed sufficiently high on the valley slope so that it is above the ultimate height of the WRSF so that construction of the latter would not interfere with the tailings operation.

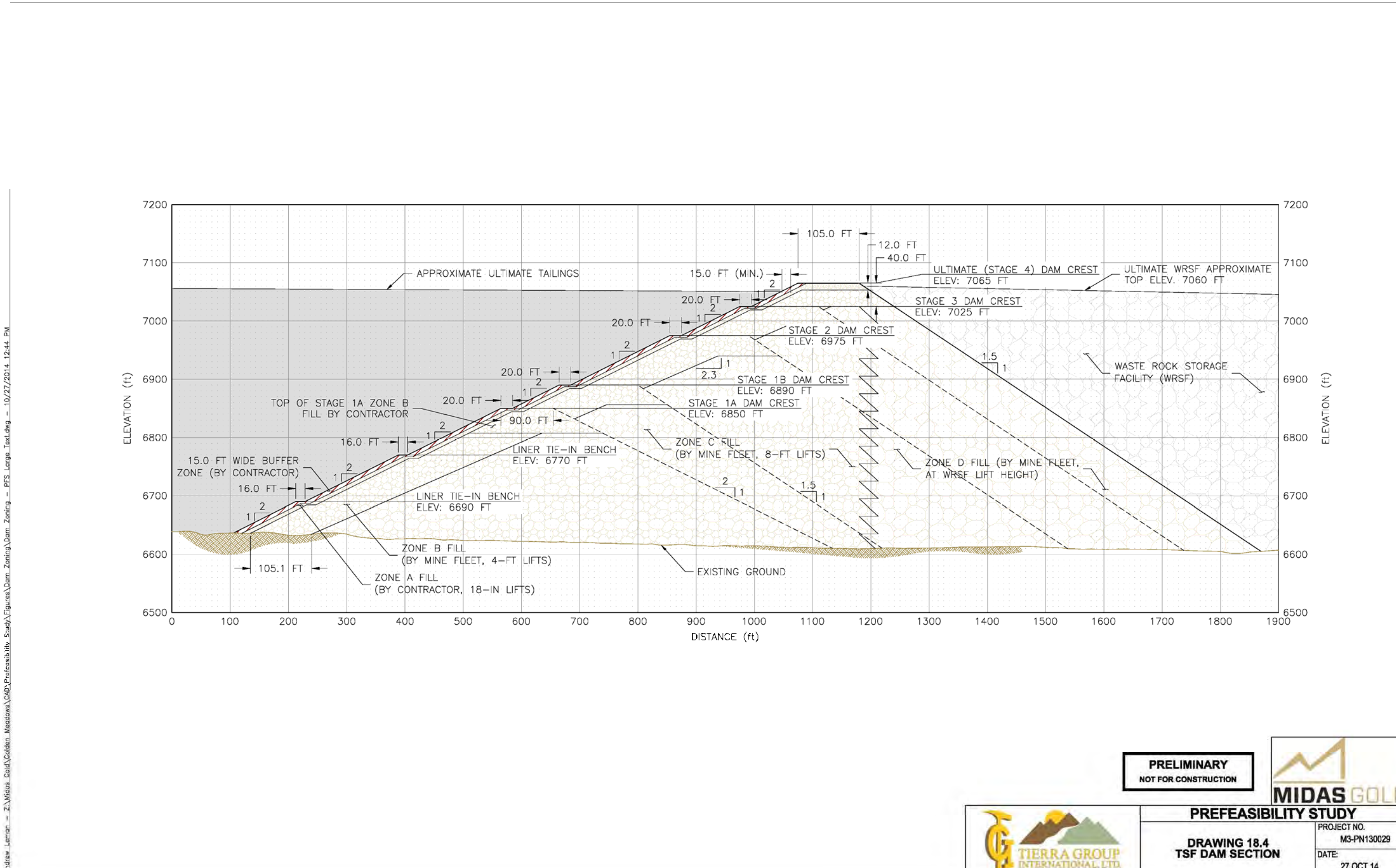
18.12.3 Waste Rock Storage Facility

The main WRSF would be located immediately east of the TSF, between the TSF and the Hangar Flats open pit. It would receive waste rock and overburden from mining the Yellow Pine and Hangar Flats open pits, totaling approximately 149 million short tons, in addition to that placed as rockfill for TSF dam construction. Most of the waste rock from the West End open pit (approximately 105 million short tons) would be used to backfill portions of the West End and Yellow Pine pits, with the remainder (approximately 25 million short tons) stored at the West End WRSF. With SODA material included, the TSF dam and WRSF combined would hold approximately 210 million short tons of waste rock and overburden.

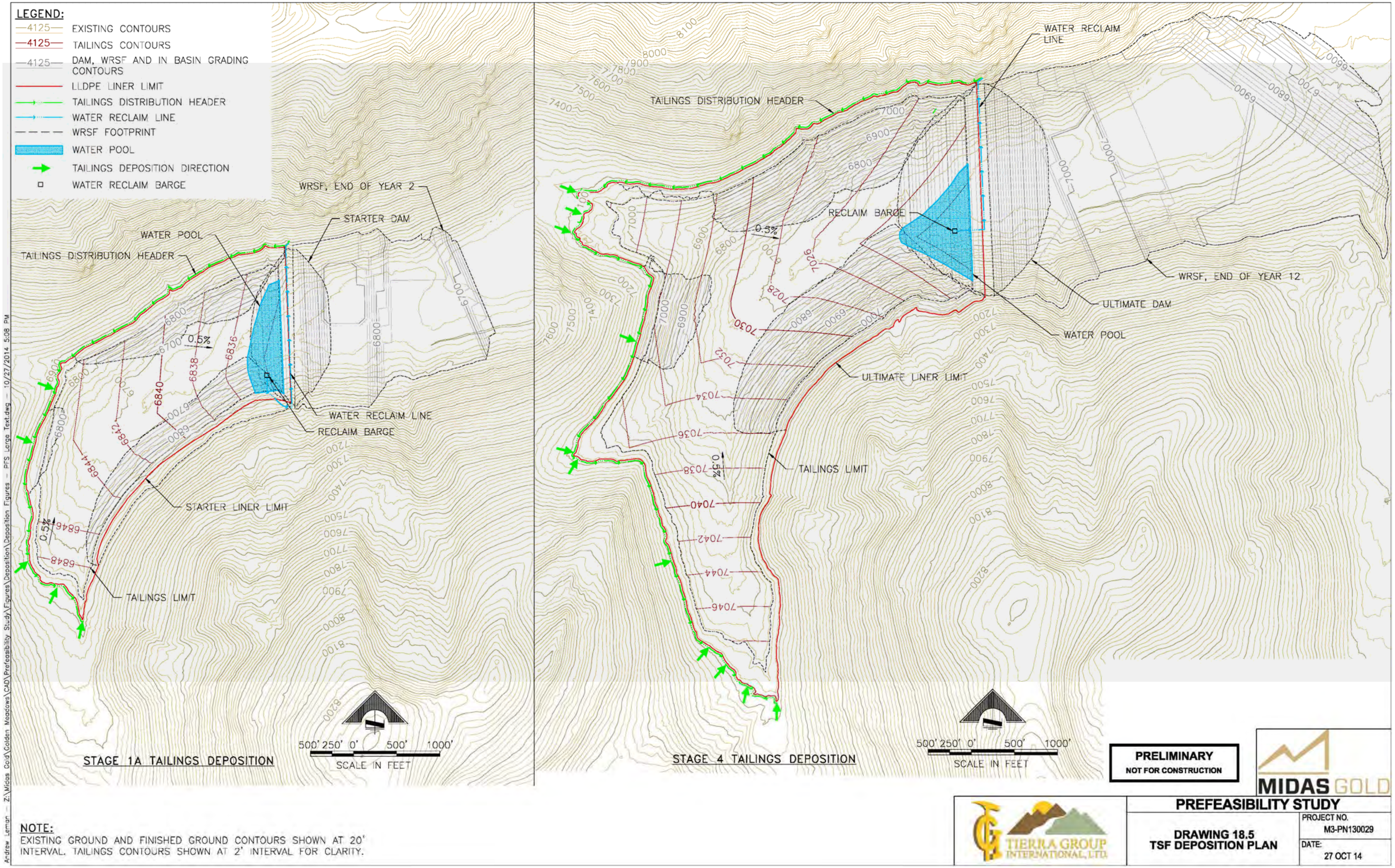
The initial lift of the WRSF would be placed at the toe of the TSF dam, with the WRSF expanding vertically and downstream as waste placement progresses. The WRSF would thus provide a continuously-growing buttress for the TSF dam, significantly enhancing dam stability and eventually reaching a thickness of 0.6 miles.

If the results of geochemical testing (currently in-progress) indicate the need for special handling of certain waste materials, a waste management / placement plan would be developed. Waste with higher metal-leaching or acid generation potential would either be blended with neutralizing material such as that from West End pit, or segregated in a location within the WRSF that minimizes potential exposure to air and moisture.

As discussed in Section 18.8, runoff and seepage would be collected at the toe of the WRSF using berms and ditches, and routed to ponds for settling of sediments and potential reuse. Collection ditches and diversion berms would be rebuilt at the toe of the WRSF as it expands, minimizing the commingling of contact and non-contact water. Drawing 18.1 shows the water management plan for the WRSF and TSF.



Drawing 18.4: TSF Dam Cross-Section



Drawing 18.5: TSF Deposition Plan

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19 MARKET STUDIES AND CONTRACTS

19.1 MARKET STUDIES

19.1.1 Doré

The economic analysis completed for this PFS assumed that gold and silver production in the form of doré could be readily sold without deleterious element penalties. Assumed gold and silver doré payabilities, refining and transport charges are provided in Table 19.1; these values are considered typical.

Table 19.1: Dore Payables, Refining and Transportation Assumptions

Parameter	Gold in Doré	Silver in Doré
Metal Payability in Doré	99.5%	98.0%
Refining Charges	$1.00/oz Au	$0.50/oz Ag
Transportation Charges	$1.15/oz Au	$1.15/oz Ag

19.1.2 Antimony Concentrate

A preliminary market study for the sale of antimony concentrate was completed by a confidential independent leading industry participant. The marketing study was based on preliminary antimony concentrate production estimates, ranges for projected antimony, gold, silver, and deleterious element grades in the concentrate. The following information was derived from the antimony market study:

- Approximately 200,000 tonnes of antimony is presently produced annually around the world. One quarter of the production is from recycling while the remaining three-quarters result from primary production.
- The antimony concentrate production profile of this Project, based on the mine plan provided in Section 16, would make it one of the largest antimony producers outside of Asia.
- Antimony concentrate payables would potentially be:
  - 60 to 70% payable for an antimony concentrate with a grade of 55 to 60% antimony, respectively, with no treatment or refining charges and no minimum deductions;
  - deleterious element charges may apply, particularly for selenium and arsenic;
  - gold would not be subject to refining or other deductions and would yield payables of:
    - 15 to 20% for concentrate gold grades of 5.0 to 8.5 g/t Au, respectively;
    - 20 to 25% for concentrate gold grades 8.5 to 10.0 g/t Au, respectively; and
    - 25% for concentrate gold grades greater than 10.0 g/t Au.
  - silver would not be subject to refining or other deductions and would yield payables of:
    - 40 to 50% for concentrate silver grades of 300 to 700 g/t, respectively; and
    - 50% for concentrate silver grades greater than 700 g/t.
- Currently only a small number of smelters, all of them located in Asia, have the capacity to treat the volume of antimony concentrate planned for production by the Project. Other smelting possibilities outside of Asia

were discussed, but such facilities are only at the planning stage, and may or may not be viable alternatives 5 to 7 years in the future.

Based on the payability information provided by an independent leading industry participant, and on the concentrate transportation costs discussed in Section 18, Table 19.2 summarizes the antimony concentrate payables and transportation charge assumptions for this study.

Table 19.2: Antimony Concentrate Payables and Transportation Assumptions

Parameter	Concentrate Payables and Transportation Charges
Antimony Payability	Constant at 68% (based on a constant life-of-mine concentrate grade of 59%)
Gold Payability	<5.0 g/t Au no payability ≥5.0 g/t ≤8.5 g/t Au payability of approximately 15 - 20% ≥8.5 g/t ≤10.0 g/t Au payability of approximately 20 - 25% ≥10.0 g/t Au payability of approximately 25%
Silver Payability	<300 g/t Ag no payability ≥300 g/t ≤700 g/t Ag payability of approximately 40 - 50% ≥700 g/t Ag payability of approximately 50%
Transportation Charges	$151/wet tonne from site to Asia

19.2 METAL PRICES

The metal prices selected for the four economic cases in this report are shown in Table 19.3; the basis for selection of these metal prices is also provided in the table.

Table 19.3: Assumed Metal Prices by Case

Case	Metal Prices			Basis
	Gold ($/oz)	Silver(1) ($/oz)	Antimony(1) ($/lb)	
Case A	$1,200	$20.00	$4.00	Lower-bound case that reflects the lower prices over the past 36 months and spot on December 1, 2014.
Case B (Base Case)	$1,350	$22.50	$4.50	Approximate 24-month trailing average gold price as of December 1, 2014.
Case C	$1,500	$25.00	$5.00	Approximate 48-month trailing average gold price as of December 1, 2014.
Case D	$1,650	$27.50	$5.50	An upside case to show Project potential at metal prices approximately 20% higher than the base case.

Note:
(1) Prices were set at a constant gold:silver ratio ($/oz:$/oz) of 60:1 and a constant gold:antimony ratio ($/oz:$/lb) of 300:1 for simplicity of analysis, although individual price relationships may not be as directly correlated over time. Historic gold:silver ratios have averaged around 60:1.

There is no guarantee that the gold, silver, and antimony prices used in the study cases would be realized at the time of production. Prices could vary significantly higher or lower with a corresponding impact on Project economics. Historical gold, silver and antimony prices, shown on Figure 19.1, Figure 19.2 and Figure 19.3, respectively, highlight the variable nature of metal prices and their recent historical high level.

Figure 19.1: Historical LME Gold PM Fixed Prices

Figure 19.2: Historical LME Silver PM Fixed Prices

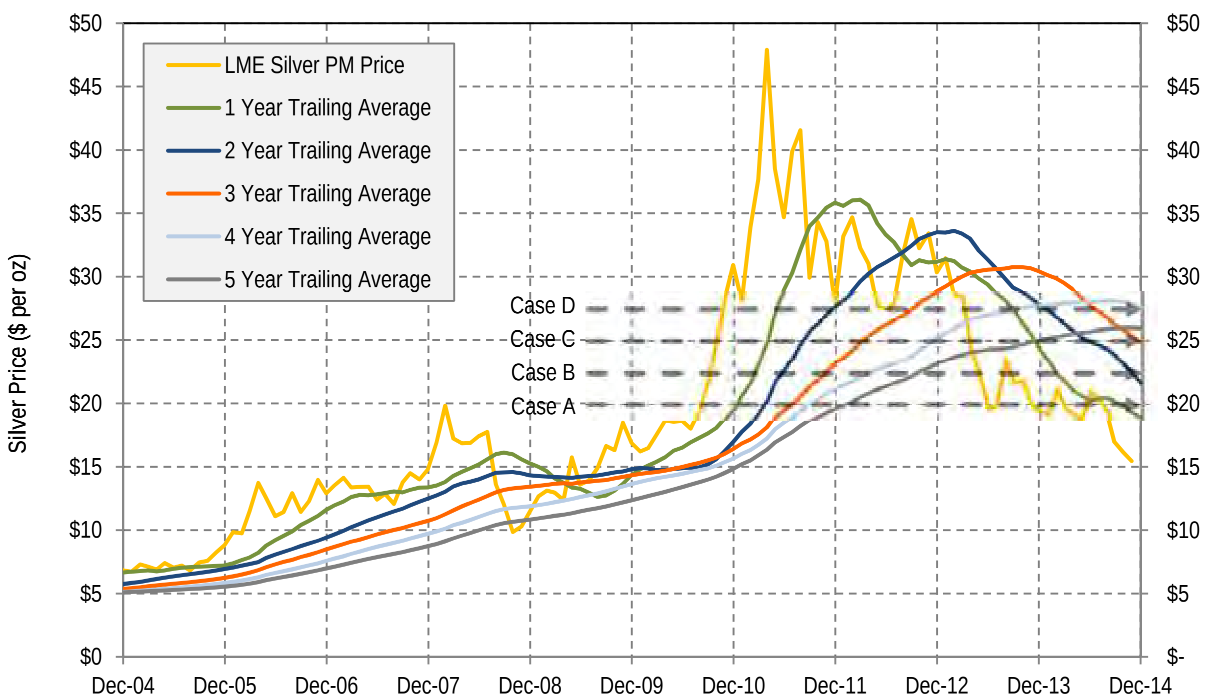
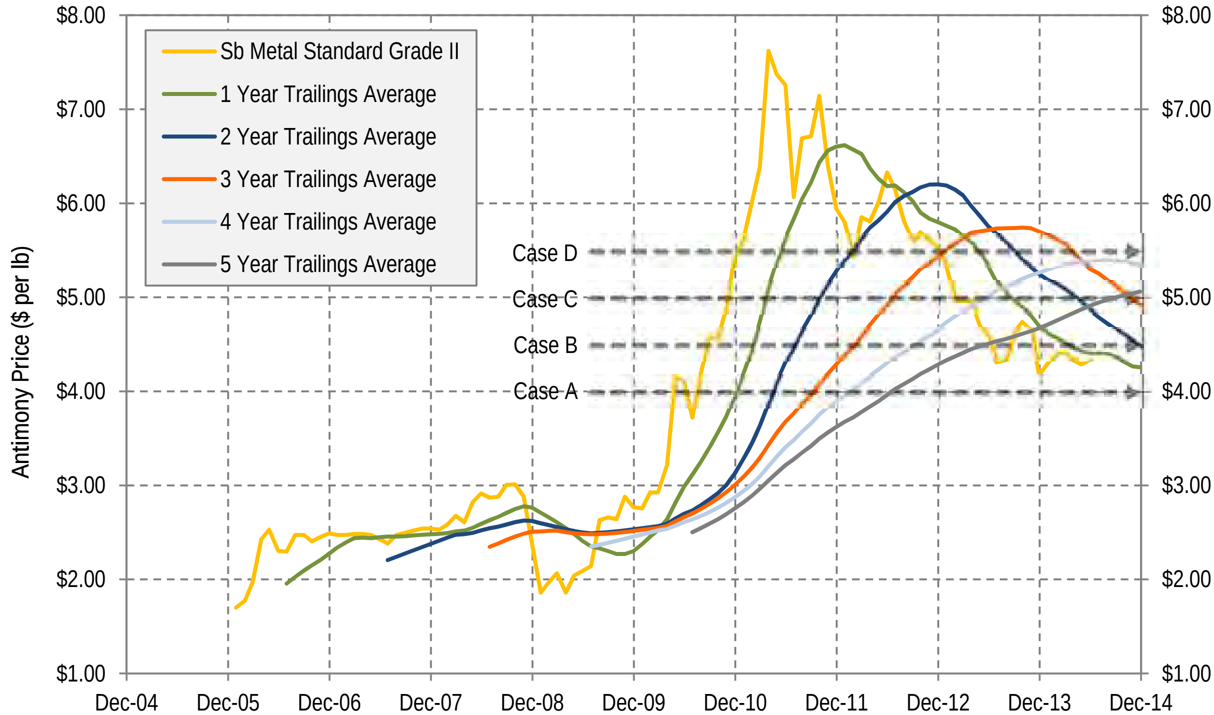


Figure 19.3: Historical Antimony Metal Standard Grade II Prices



19.3 CONTRACTS

There are no mining, concentrating, smelting, refining, transportation, handling, sales and hedging, forward sales contracts, or arrangements for the Project. This situation is typical of an exploration project that is still several years away from production.

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20 ENVIRONMENTAL STUDIES PERMITTING AND SOCIAL COMMUNITY IMPACTS

In conjunction with its redevelopment of the Stibnite Gold Project, Midas Gold would restore much of the historically impacted brownfields site to a more natural condition than exists today. The Project area has been mined extensively for tungsten, antimony, mercury, gold, and silver since the early 1900s in numerous episodes historically and in the modern era. The District has a strong history of providing strategic metals to the United States during times of high demand such as war time critical minerals shortages; providing substantial economic benefit to the local counties and the State of Idaho, and providing much needed jobs and support to local businesses for nearly 100 years. These various historic mining efforts have shaped the District, changed the paths of rivers and streams, and left significant legacy impacts. The significant history of this District has left a variety of Recognized Environmental Conditions (RECs), legacy environmental features, areas of significant disturbance, and residual surface features that persist to this day.

Multiple cleanup efforts undertaken at the site by Federal and State agencies and private companies pursuant to multiple cooperative agreements include stream improvements, historic tailings reclamation efforts, facility removal and cleanup, surface disturbance reclamation, and specific action cleanup projects under the CERCLA. These projects have improved water quality, isolated historic waste features, and improved sediment control from disturbed ground; however, there still remain substantial surface disturbances, major sediment sources, water quality impacts, and degraded aquatic and terrestrial wildlife conditions, compounded further by extensive forest fire impacts and subsequent damage from soil erosion, landslides and debris flow, and resultant sediment transport.

In the Stibnite Gold Project design, Midas Gold has created a plan to restore much of the site by removing existing barriers to fish migration and re-establishing salmon and steelhead fish passage, removing uncontained historic tailings, reusing historic spent ore material for construction, restoring stream channels, and implementing sediment control projects such as on the East Fork of Meadow Creek (aka Blowout Creek). In addition to remediating historic disturbance, Midas Gold has endeavoured to minimize the Project's footprint and related impacts by siting facilities and roads on previously disturbed ground and away from riparian areas. Midas Gold has designed the major access route into the Project site to include existing roads for most of its length, with some alternate sections designed to avoid rivers and large waterways, resulting in maximum sediment control and related protection of water quality, and reducing risks of vehicle incidents impacting waterways. By including some of the workforce in an office complex in the nearby town of Cascade, the on-site workforce (the majority of which will be bussed in) would be minimized, resulting in less traffic. Re-establishment of historic electrical line power to the Project site will serve to minimize fossil fuel consumption and related haulage along the access route, and improve the reliability of services for communities and residents along the power-line corridor.

The following sections provide Project-related information on site characterization efforts and existing conditions, anticipated permitting requirements for potential development, social and community impacts and considerations, geochemical materials characterization, mitigation of stream and wetland potential disturbances, and reclamation and closure.

20.1 ENVIRONMENTAL BASELINE STUDIES

An extensive set of baseline data demonstrating historic and existing conditions exists for the Stibnite Gold Project site. Midas Gold contracted HDR, Inc. (HDR) and MWH Americas, Inc. (MWH) to provide an environmental adequacy review of all available environmental baseline reports and data compiled for the period 1979 through present. This adequacy review supplements information developed for two prior Environmental Impact Statements completed for the site since 1982 that were conducted in connection with historical mining operations discussed in Section 6 of this Report. The adequacy review was the basis for the design of individual work plans for collection of environmental baseline data. These work plans are discussed later in Section 20.1.4.

20.1.1 USFS/EPA Stibnite Characterization and Risk Assessment

An environmental site characterization was conducted at the Project site from 1998 through 2000 by the USFS and the EPA (who contracted URS Corporation (URS) to complete the work). The Stibnite Site Characterization and Risk Assessment (URS, 2000b) (the "URS Report"), involved a complete site characterization that addressed:

- geology and hydrology (surface water and ground water);
- surface water features and stream classification;
- fish and wildlife;
- aquatic resources;
- vegetation;
- air quality;
- land use; and
- human health.

The chemical, biological, and habitat characterization presented in the URS Report was used to prepare a risk evaluation report that determined there were no unacceptable risks to the environment or human health posed by chemical or physical stressors described in the URS Report. For all categories of populations exposed, the risk was shown as "unlikely", and there were no populations (fish, wildlife, or human) shown as having a "likely" risk. This all-inclusive environmental baseline has been and is being augmented by additional Project environmental baseline studies as described in the following sections.

20.1.2 MSE Environmental Site Assessments

In 2009 and 2010, Midas Gold and Vista US contracted Millennium Science & Engineering, Inc. (MSE) to conduct Phase I and Phase II Environmental Site Assessments (ESAs), as prescribed by the American Society for Testing and Materials (ASTM) Standard Practices for Environmental Site Assessments (E 1527-05) (ESAs) and ASTM Standard Guide for Site Assessments; and Phase II Site Assessment Process (E-1903-97) for multiple parcels within the Project area; additional parcels acquired in April and May 2011 that were not part of the ESA's were assessed by Midas Gold but have not been the subject of any physical disturbance by Midas Gold.

The purpose of the Phase I ESA was to identify, pursuant to the processes prescribed in Standard Practice E 1527-05, RECs in connection with the Property. The term "recognized environmental condition" is defined by ASTM as "The presence or likely presence of any hazardous substances or petroleum products on a property under conditions that indicate an existing release, a past release, or a material threat of a release of any hazardous substances or petroleum products into structures on the property or into the ground, groundwater, or surface water of the Property." The extent and coverage of the MSE investigation follows ESA standards developed by ASTM as set forth in Standard Practice E 1527-05 (ASTM 2005) and the All Appropriate Inquiry (AAI) standard in Title 40 Code of Federal Regulations (CFR) Part 312. The ASTM Standard E1527-05 defines good commercial and customary practice for conducting an ESA of a parcel of commercial real estate with respect to the range of hazardous materials (within the scope of CERCLA and potential presence of petroleum products or residual which could adversely affect the environment. As such, this practice is intended to enable a user to satisfy, where applicable, the bona fide prospective purchaser (BFPP), contiguous property owner (CPO), and the "innocent landowner" defenses.

The Phase II investigation by MSE was conducted in coordination with a Phase I ESA of the Property also prepared by MSE. As noted above, the primary purpose of the Phase I ESA was to identify RECs associated with a specific site using information from limited sources such as literature reviews, public records, personal interviews, and a simple verification site visit. The purpose of the Phase II Environmental Analysis and Review was to further evaluate several of the significant RECs identified in the Phase I, conduct a regulatory review of likely applicable environmental regulations, further review existing data, and perform field verification and collection of additional site specific data. For this investigation by MSE, the further evaluation of significant RECs and data analysis/collection was limited to the following areas:

- geomorphic stability;
- aquatic and riparian ecology/fisheries;
- surface water quality;
- slope stability; and
- air quality.

The primary purpose of a Phase II ESA as detailed in "ASTM E1903 – 97 (2002) Standard Guide for Environmental Site Assessments: Phase II Environmental Site Assessment Process" is "to evaluate the recognized environmental conditions identified in the Phase I ESA or transaction screen process for the purpose of providing sufficient information regarding the nature and extent of contamination to assist in making informed business decisions about the property; and where applicable, providing the level of knowledge necessary to satisfy the BFPP, CPO, and innocent purchaser defenses".

The results of the ESAs indicate that overall water quality in all drainages is marginally impaired due to the highly mineralized nature of area and the duration and extent of historic mining. However, a site characterization conducted in 2000 by URS showed that surface water quality in the Meadow Creek and the EFSFSR improved substantially between 1997 and 1999 as a result of the Bradley Tailings Diversion and Reclamation Project.

There were 88 potential or known RECs in the evaluated portion of the Property that were categorized based on their risk rating as defined in Table 20.1.

Table 20.1: Recognized Environmental Conditions by Category

REC Category	Number	Description of Category of REC
Critical	0	Imminent threats to human health or the environment
Significant	15	High volume of waste or potential for high contaminant concentrations
Moderate	40	Moderate volume of waste, footprint or potential contaminant concentrations
Low	33	Low or unlikely to impact surface or groundwater

Of the 88 documented RECs, none are in the "Critical" category according to the MSE Report. Midas Gold has since amalgamated many of these RECs and reduced the number of identified potential RECs to 24; examples include amalgamating into a single REC those being counted as two or more due to continuity across a claim boundary, such as was the case with the SODA being counted independently on both patented and unpatented land. Six of these 24 potential RECs are located solely on patented land; four are located solely on unpatented land; and 14 of the potential RECs are located on both patented and unpatented land. Table 20.2 presents a consolidated REC summary.

Table 20.2: Consolidated RECs by Category and Ownership

REC No.	REC Description	Patented / Unpatented	MSE Risk Category
1	Spent Ore Disposal Area (SODA)	Both	Low - High
2	Former Meadow Creek Mill and Smelter Complex	Both	High
3	Hecla Heap and Former Processing Facilities	Patented	Low - Medium
4	Wastewater land application of heap leach effluent	Both	Low
5	Former Stibnite Mines Inc. (SMI) Heap Leach Pads	Patented	Medium
6	Former SMI Processing Plant	Patented	Low - Medium
7	Former Fuel Oil ASTs Areas	Patented	Medium
8	West End (Waste Rock)	Both	Medium - High
9	Yellow Pine Pit (Historic Tailings)	Patented	High
10	Yellow Pine Pit Waste Rock	Both	Medium - High
11	Former Monday Camp	Both	Medium
12	USFS Yellow Pine Repository	Patented	Low
13	Former Barrel Dumps	Both	Low
14	DMEA Mine Waste Rock Dump	Unpatented	Medium
15	Former Stibnite Landfill	Unpatented	Low - Medium
16	Former Pilot Plant	Unpatented	Low
17	Former Stibnite Service Station	Unpatented	Low
18	Historic Mine Workings	Both	Low - Medium
19	General Erosion and Sediment Transport	Both	Low - High
20	Fish Migration Barrier at Yellow Pine Pit	Both	Medium
21	Contaminated Groundwater in Alluvial Aquifer	Both	High
22	Placement of Spent Ore and Waste Rock Throughout District	Both	Medium
23	Slope Stability / Geotechnical	Both	Low - Medium
24	Surface Water Quality	Both	Medium

As noted above, none of the RECs were deemed “Critical” and no RECs were categorized as imminent threats to human health or the environment. Midas Gold is actively monitoring surface water, groundwater, seeps and springs at the site, according to the approved monitoring plan. The United States Geologic Survey (USGS) continuously monitored water quality on the site between from 1983 to 1996, and, in addition to Midas Gold’s water quality monitoring; the USGS began its water quality monitoring again in 2011 through a partially funded cooperative arrangement with Midas Gold. These continuing monitoring programs serve as environmental protections, collection of important environmental baseline data and maintenance of Midas Gold’s BFPP status. Midas Gold intends to avoid causing disturbance of existing RECs unless necessary in the context of exploration or potential future development of the Project. At this time, based on known RECs, Midas Gold is not aware of any existing risks that could materially affect potential development at the Project site.

Although some portions of the Project site were placed on the Federal Facilities Docket on September 25, 1991, and are currently listed on the Comprehensive Environmental Response, Compensation, and Liability Information System (CERCLIS) List (No. ID9122307607), in 2001 both the EPA and the Bureau of Environmental Health and Safety (BEHS), Division of Health, Idaho Department of Health and Welfare determined the risk to be too low for listing on the National Priorities List (NPL). No further public action by the EPA or Idaho Department of Environmental Quality (IDEQ) has been pursued.

20.1.3 Consent Decrees under CERCLA

Several of the patented lode and mill site claims acquired by Midas Gold comprising part of the West End Deposit, and the Cinnabar claims held under purchase option from the Estate of J.J. Oberbillig are the subject of a consent decree entered in the United States District Court for the District of Idaho (United States v. Estate of J.J. Oberbillig, No. CV 02-451-S-LMB (D. Idaho)) in 2003. Portions of the Yellow Pine and Hangar Flats deposits are the subject of a consent decree entered in the United States District Court for the Northern District of California (United States v. Bradley Mining Company, et al., No.CV-03968 TEH) in 2012. These consent decrees which Midas Gold has discussed with the relevant regulatory agencies, involve or pertain to environmental liability and remediation responsibilities with respect to the affected properties described in each. Among and subject to the various provisions in each of the decrees, these decrees can be generally described as providing the regulatory agencies that were party to the agreement access the right to conduct remediation activities and also requiring that successors and assignees refrain from activities that would interfere with or adversely affect the integrity of any remedial measures implemented by government agencies.

Midas Gold has taken all reasonable steps to locate but cannot ensure it has identified every consent decree or administrative order which may affect the Project site. In addition, the EPA, the Forest Service, and the State of Idaho have jointly identified certain required environmental remediation measures for the affected mineral properties, some of which comprise a portion of the historical Stibnite mine site, pursuant to CERCLA and the Resource Conservation and Recovery Act (RCRA). The Bradley and Oberbillig decrees and any required remediation measures specified by these agencies may impact future exploration and development activities on the affected properties. In addition to such required measures, Midas Gold has undertaken and plans to further undertake voluntary remediation measures to improve the overall environmental condition of the Stibnite Gold Project area. Existing consent decrees do not preclude Midas Gold from pursuing these steps to remediate and improve the environmental status of the site. These steps are described in more detail later in this Report.

20.1.4 HDR and MWH Adequacy Audits

In 2011, Midas Gold retained environmental consulting firms HDR and MWH of Boise, Idaho, and their sub-consultants, to conduct technical adequacy audits of all existing environmental information, and to develop individual work plans to conduct an environmental baseline collection program. These workplans were developed to include the following resource listing:

- aquatic resources;
- air quality;
- cultural resources;
- environmental justice;
- geochemistry;
- soils and geology;
- groundwater hydrology;
- groundwater quality;
- noise;
- public health and safety;
- recreation;
- socioeconomics;

- surface water hydrology;
- surface water quality;
- terrestrial vegetation;
- terrestrial wildlife;
- transportation and site access;
- visual resources;
- water rights; and
- wetland resources.

The environmental baseline work plans for these subject categories prepared by HDR and MWH have been approved by the USFS interdisciplinary team (ID Team), with input from involved state and federal agencies. The ID Team was specifically organized to oversee these environmental studies, to establish the existing environmental conditions, identify and quantify environmental risks and liabilities, and monitor for potential impacts from onsite activities. The ID Team is comprised of highly qualified specialists in each of the resource categories. Initial supplemental baseline studies in the areas of surface and ground water, wetlands, and vegetation were initiated in the summer of 2011. Geotechnical and geochemical fieldwork also commenced during the 2011 field season, as were fisheries, wildlife, transportation and other needed baseline studies. In early 2013, Midas Gold contracted senior NEPA specialists to conduct a baseline adequacy review to increase assurance that appropriate baseline studies were being undertaken to support future NEPA analysis of a potential mining project.

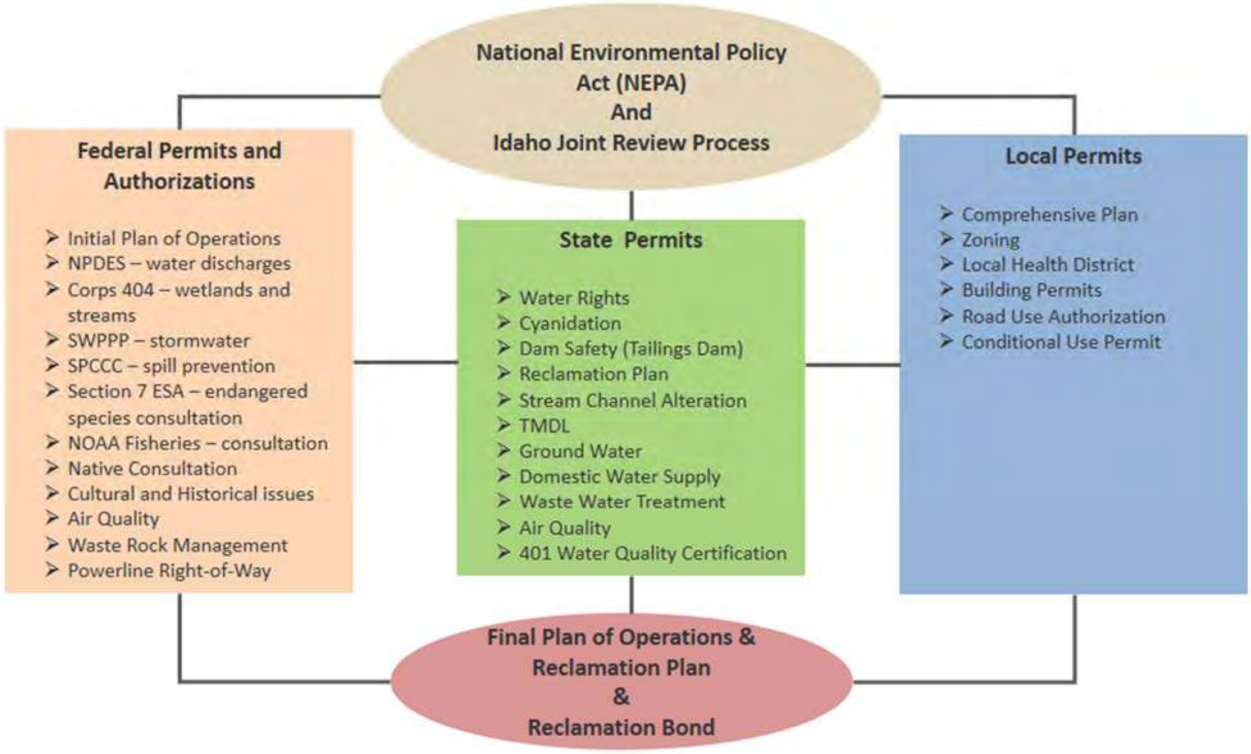
The environmental baseline program for all the major resource categories would continue through 2015 in order to accurately describe the existing environment at the “brownfield site”, and allow for a "full and fair" discussion of all potentially significant environmental impacts in the event that the Stibnite Gold Project moves forward. The fact that the entire Stibnite Gold Project has been studied extensively, both historically and currently, ensures the scientific integrity of the methodologies and analysis used to collect the data; this ensures that a meaningful analysis can conducted, allowing for a comparative assessment of all alternatives.

20.2 PERMITTING

20.2.1 Environmental Impact Statement

USFS approval of any Final Plan of Operations (PoO) / Reclamation Plan for the Project requires an environmental analysis under NEPA. NEPA generally requires federal agencies to study and consider the likely environmental impacts of the proposed action before taking whatever discretionary federal action is necessary for the Project to proceed. Figure 20.1 shows the "umbrella" structure of NEPA.

Figure 20.1: Mine Permitting Requirements



Under NEPA, the "purpose and need" for the potential Project would be to conduct open pit mining, which would disturb approximately 1,425 acres of land on unpatented and patented mining claims within the Project area to produce gold, antimony, and other minerals from mineralized material reserves, of which approximately 680 acres is deemed already disturbed or impacted by prior mining-related activities. The Project mining operations would provide the opportunity to improve the environmental condition of previously disturbed and surrounding site, as described elsewhere.

The EIS and the related Record of Decision (ROD) for PoO approval serves as an "overarching" procedural permitting requirement, as well as that of at least three other primary federal authorizations or determinations:

- National Pollutant Discharge Elimination System (NPDES) Permit for water discharge;
- United States Corps of Engineers (USACE) 404 Dredge and Fill Permit; and
- Endangered Species Act (ESA) Biological Opinion.

The EIS and ROD for the PoO effectively drive the entire permitting process, since a completed final EIS and favorable ROD are generally required before these important clearances can be obtained or utilized.

The Council on Environmental Quality identifies 10 factors for determining the significance of a proposed action, and the potential requirement for an EIS. Of these, three primary circumstances are related to mining activities:

- potential impacts on wilderness and other pristine, undeveloped areas;
- potential impacts on threatened and endangered species; and
- situations where several individual mining projects would affect a single watershed circumstance.

The fact that the Project site is a "brownfield site" (extensively mined in the past) and the subject of historic CERCLA cleanup actions, along with potential impacts on threatened and endangered species, would combine to require an EIS for any proposed action for developing a mine and processing facilities at the site.

Other primary federal and state authorizations and/or permits are described in the sections which follow. The discussion ties EIS and other permitting requirements together in terms of an estimated schedule and costs for completing the program.

20.2.2 National Pollutant Discharge Elimination System Permit

An NPDES Permit is required for point source discharges from the mining operation to "waters of the United States". In addition, since the Project is subject to performance standards for “new sources” for its respective industrial source category, the Project must demonstrate that it is applying the best available control technology to meet applicable water quality standards. The permit application must be submitted at least 180 days prior to the approved discharge.

Storm water discharges associated with this industrial activity require a related permit. Storm water is defined as "storm water runoff, snowmelt runoff, and surface runoff and drainage". Active storm water would be managed via a storm-water pollution prevention plan (SWPPP). This document must also be submitted at least 60 days before commencing the discharge. Where flows are from conveyances that are not impacted by operational activities, or do not come in contact with overburden or other mine waste, a permit is not required. Hence, the water management scheme developed for the Project endeavours to collect and convey clean water around the mining operation and discharge downstream, wherever feasible and practicable.

20.2.3 U.S. Army Corps of Engineers Section 404 Dredge and Fill Permit

A Section 404 Permit is required under the Clean Water Act for the discharge of dredged or fill material placed into waters of the United States. Dredged or fill material includes tailings and waste rock. Other activities, in addition to the tailings and waste rock storage that may require a 404 Permit are:

- road construction;
- bridges;
- construction of dams for water storage;
- stream diversions; and
- certain reclamation activities.

Waters of the United States include certain defined wetlands. A 2009 U.S. Supreme Court decision found mine tailings to be "fill", and can, therefore, be placed into waters of the United States with an approved Section 404 USACE Dredge and Fill Permit.

20.2.4 ESA Consultation

The Endangered Species Act prohibits the taking of fish and wildlife species classified as endangered or threatened, unless otherwise authorized. The following species are, or may be, in the vicinity of the Project site:

- spring/summer Chinook salmon (threatened);
- steelhead trout (threatened);
- bull trout (threatened);
- west slope cutthroat trout (sensitive);

- Canada lynx (threatened);
- whitebark pine (sensitive); and
- milkvetch (sensitive).

A biological opinion would be required for the listed species, under Section 7 of the Clean Water Act. Federal agencies are required to "conserve endangered or threatened species, and to ensure that their actions are not likely to jeopardize the continued existence of any of these species or adversely modify their designated habitat" (ESA, 16 U.S.C. Section 1538(a). Consultation with National Oceanic and Atmospheric Administration (NOAA) Fisheries for Chinook salmon and steelhead trout species would be required. The United States Fish and Wildlife Service (USFWS) would be the consulting agency for bull trout (non-anadromous fish) and the Canada lynx.

The consultation process would be run concurrently with the EIS. Some adverse effect is allowed, provided it does not jeopardize the continued existence of the species. Typically, a biological assessment (precursor to biological opinion) is prepared by the third-party contractor preparing the EIS. This assessment can be used to satisfy both the requirements of the ESA and NEPA. If the USFS concludes that the Project may affect a listed species or habitats, the assessment would then require formal consultation and a biological opinion. This involves:

- a summary of the information upon which the USFWS' opinion is based;
- a detailed discussion of the effects of the actions on listed species or critical habitat; and
- USFWS' opinion as to whether the agency action would jeopardize "the continued existence of the species, or adversely modify their critical habitat". The formal biological opinion must be issued within 135 days from the date that the formal consultation is initiated.

20.2.5 Other Federal Programs

There is no comprehensive federal groundwater quality statute, in contrast to surface water and the Clean Water Act. Ground water protection is found in several programs which include: the Safe Drinking Water Act, sections of CERCLA, and the RCRA. The Safe Drinking Water Act was implemented by the State of Idaho to enforce drinking water regulations for municipalities, public water systems, and related facilities. Based on the anticipated number of personnel, this operation would be classified as a public water system.

The federal Clean Air Act regulates air quality and the Project would be subject to National Ambient Air Quality Standards; definitive air quality criteria would apply. The operation would be required to meet Prevention of Significant Deterioration requirements, visibility regulations, and National Emission Standards for Hazardous Air Pollutants. This would involve pre-construction and operating permits issued and managed by the State of Idaho described below in this section.

20.2.6 Major State Authorizations, Licences, and Permits

The federal and state application processes would be integrated and processed concurrent with the EIS. The key authorizations, licenses, and permits required by the State of Idaho are as follows:

- Air Quality Application for Permit to Construct and Operate – This permit assesses the allowable impacts to air quality, and prescribes measures and controls to reduce and/or mitigate impacts.
- Cyanidation Permit – This permit is required by IDEQ and is applicable for a facility that processes mineralized material using cyanide as the primary reagent. Midas Gold intends to produce gold doré onsite and uses cyanide in its production. The regulations apply to both operations and closure and reclamation of any cyanide facility.

- Land Application Permit – In order to apply any treated process wastewater to a designated land area for ultimate disposal, the mining company must obtain a Land Application Permit from IDEQ. This could be required in order to meet the performance standards for new sources "zero discharge" requirement for net precipitation minus evaporation. This is also a safeguard to ensure no unpermitted discharges.
- Ground Water Rule – This rule establishes minimum requirements for ground water protection through standards and a set of aquifer protection categories. To implement the rule, Midas Gold would need to request the establishment of points of compliance outside and down-gradient from the mine area(s). Midas Gold would also establish reasonable upper-tolerance limits for all compliance wells, working directly with IDEQ. These upper-tolerance limits would take into account the high naturally-occurring background levels for several parameters.
- Total Maximum Daily Loads (TMDL) – In Idaho, TMDLs are generally assessed on a sub-basin level, which means water bodies and pollutants within a hydrologic sub-basin are generally addressed within a sub-basin report. An earlier TMDL for the main-stem South Fork Salmon River was approved by EPA in 1991. That TMDL set surrogate sediment targets for percent fines and cobble embeddedness. The Salmon River, South Fork Sub-basin report was updated in 2012 with an EPA approved addendum in February 2012 that proposed to remove the EFSFSR from the 303(d) list for sediments and metals. Recent reclamation work has stabilized historic mine and mill tailings in the discharge, and reduced transport of metals in sediment (HDR, 2012). This program of concurrent reclamation, best management practices (BMP) applications, and special environmental enhancement projects would be continued by Midas Gold during construction and over the life of the Project, should it proceed.
- Water Rights – As described in Section 5 of this technical report Midas Gold currently holds four permanent water rights associated with the mining activity area. Additional water rights will need to be secured through direct permit application and subsequent approval of such rights from the Idaho Department of Water Resources (IDWR) in order to have sufficient water rights to support Project development.
- Stream Channel Alteration Permit – This permit is required by the IDWR for a modification, alteration, or relocation of any stream channel within or below the mean high water mark. The PFS contemplates relocating Meadow Creek and the EFSFSR, both temporarily and permanently, as part of the overall mine plan. This permit would be obtained in conjunction with any USACE 404 permit obtained for the same purpose.
- Dam Safety Permit – The IDWR requires a Dam Safety Permit for dams greater than 10 ft high or for reservoirs exceeding a 50-acre-feet storage capacity. The Application to Construct a Dam includes design plans and specifications for construction of the dam. Mine tailings impoundments greater than or equal to 30 ft high are regulated by IDWR in the same manner. Design and construction requirements for mine tailings impoundment structures are described in IDAPA 37.03.05. The PFS contemplates construction of a TSF in the Meadow Creek drainage and would need to seek to obtain this authorization.
- Water and Wastewater Systems – The drinking water system(s) design for the contemplated work camp (construction and operations) must be approved prior to use. This would assure compliance with the Safe Drinking Water Act. IDEQ would also require approval of plans and specifications for any new sewage treatment and disposal for the work camp.
- Fuel Storage Facilities – Any proposed fuel storage must also comply with IDEQ design and operating standards, as well as Idaho State Fire Marshall and Valley County requirements.
- Reclamation Plan – All surface mines must submit and obtain approval of a comprehensive reclamation plan (Title 47) for mining activities on patented land as administered by the Idaho Department of Lands (IDL). This includes detailed operating plans showing pits, mineral stockpiles, overburdened piles, tailings ponds, haul roads, and all related facilities. The Reclamation Plan must also address appropriate BMPs,

and provide for financial assurance in the amount necessary to reclaim those mining activities. The plan must be approved prior to any surface disturbance. A large portion of the contemplated Yellow Pine, West End, and Hangar Flats pits and associated facilities are located on patented land.

- State Historic Preservation Office – Approval of a historic/cultural resources assessment by the State Historic Preservation Office would be required. The Project is located within the Stibnite National Historic District.
- Others – State requirements would also involve compliance with the Idaho Solid Waste Management Regulations and Standards, transportation safety requirements enforced by the Idaho Public Utilities Commission, and others.

20.2.7 Local County Requirements

There are several other permits and approvals that would apply to the Project including:

- conformance with the Valley County Comprehensive Plan;
- issuance of building permits by the county; and
- sewer and water systems approval by Central District Health Department, and various other authorizations.

A key annual authorization by the Valley County Road Department is the Valley County Road Use Permit for any mining operation. This permit addresses standard operating procedures for the road route to be used, seasonal limits, spill prevention and response planning, Hazardous Waste Operations and Emergency Response (HAZWOPER) or hazardous materials handling training, convoying, and other requirements.

20.2.8 Idaho Joint Review Process

The IDL is responsible for implementation of the Idaho Joint Review Process (IJRP); this process was established in order to coordinate and facilitate the overall mine permitting process in the state. The IJRP involves an interagency Memorandum of Understanding (MOU) between involved state and federal agencies. Further, the IJRP addresses a process to achieve pre-analysis coordination in approving / administering exploration permits, interagency agreement on plan completeness, alternatives considered, draft and final permits, bonding during mine plan analysis, and interagency coordination related to compliance, permit changes and reclamation/closure for major mining projects. In Idaho, the Joint Review Process was established to be the basis for interagency agreement (state, federal, and local) on all permit review requirements. The focus of the IJRP is concurrent analysis timelines; this would include, for example, in the case of Stibnite Gold Project the NEPA process, NPDES permit, USACE 404 permit, state 401 Certification of these two key permits, the State Cyanidation Permit, and the ESA Consultation. The IJRP may play a key role in achieving two primary permitting goals: (1) increased communication and cooperation between the various involved governmental agencies, and (2) reduced conflict, delay, and costs in the permitting process.

20.2.9 EIS / Permitting Sequence and Costs

This section describes the overall EIS and permitting sequence. In order to understand the sequence, recent EIS and permitting projects similar to the Stibnite Gold Project were reviewed. With regard to the likely scope of the Project, the following conceptual description was developed as the basis for this permitting analysis:

- Regulatory – EIS required; USFS Lead Agency; EPA, USACE, and IDL are cooperating agencies; National Marine Fisheries Service (NMFS) and USFWS are also possible cooperating agencies;
- Mining – estimated at 20,000 to 24,000 stpd mineralized material with an approximate 3.5:1 waste rock to mineralized material strip ratio;

- Processing – tailings by-product (commingled flotation and oxidized concentrate tailings) with high energy requirement for pressure oxidation of the mineralized material;
- Power – initial diesel generation during construction with some hydro power potential; line power would be developed to coincide with commencement of production;
- Waste Rock – potentially some selective placement would be required likely due to geochemical reactivity; large volumes would be stored and managed;
- Water Supply – available from existing or pending and future needed water rights;
- Water Treatment – appropriate water treatment technology and/or forced evaporation;
- Project Access – Burntlog route with backup provided by the South Fork and Johnson Creek roads;
- Man Camp – one camp located onsite;
- Laboratory and Office/Warehouse – located in Cascade to require EPA identification number for hazardous waste generation, storage and possible water treatment plant to pre-treat/treat laboratory wastes prior to discharge;
- Manpower – a peak construction-related workforce of approximately 1,000 direct jobs and approximately 475 to 525 direct jobs during operations;
- Operating Schedule – mining and processing year-round; and
- Total Land Disturbance– approximately 1,715 acres of patented and unpatented land (including Burntlog access route), of which approximately 684 acres is deemed already disturbed or impacted by prior activities.

This concept was developed only for the purpose of “scaling” the Project, such that the estimated schedules and costs could be compared with the projects listed earlier.

An EIS/permitting sequence is summarized below in four primary permitting windows.

1. Start baseline confirmatory studies for surface and ground water, fisheries and wildlife, geotechnical, geochemical as well as air quality and wetlands work and others. This work has been underway since 2011.
2. Commence preparation of the Initial Plan of Operations. Negotiate an MOU with the USFS for preparing the EIS. Conduct initial internal scoping with agency and political contacts concurrently, develop all other permit applications for submittal. During this period, the third-party EIS contractor would be selected by the USFS with input from the USACE and EPA. This would occur in time for the contractor to lead the scoping meetings. This assumes the EIS scoping would be conducted during this time and involves at least three public hearings (Yellow Pine, McCall, and Cascade). The contractor would then finalize the EIS work plan and initiate early environmental baseline adequacy determination write-ups for the various resource categories (air, water, socio-economics, etc.).
3. A Preliminary Draft EIS would be completed by the USFS (using a third-party contractor). This document would be for the lead and cooperating agencies and Midas Gold review only. Typically, this review would require about 90 to 120 days. In the initial stages of this period, Midas Gold would file most (if not all) of their permit applications. Some, like the water rights applications, would have already been submitted to the appropriate agencies; others, like the Corps 404 Permit and EPA NPDES Permit require the Draft EIS “preferred alternative”. The final permits cannot be issued until after the final EIS and ROD have been issued by the USFS.
4. A Draft EIS would be produced for public review; the review period would be about 60 days. The Final EIS would then be issued. At this point, the USFS could choose to issue the ROD concurrently or elect to issue

it 30 days later. There would an administrative appeal or objection period involved at this point. For the purposes of this very preliminary assessment, an additional 90 days was contemplated in this review. The remaining permits would also be issued over this period.

Estimated timelines for completion of the EIS and permits are approximately three to five years after the Plan of Operations is filed. This does not take into account time to correct potential deficiencies or the potential for objections and/or litigation and related delays, nor potential opportunities to shorten timelines based on the comprehensive work completed to characterize the site to date, and the brownfield nature of the site.

20.2.10 Midas Gold Permitting Management Strategy

To successfully achieve any such permitting program, Midas Gold has designed a seven-point management scheme that includes the following key points:

1. MOU providing for interagency cooperation, accountability, and predictability;
2. requirements for quality consultants;
3. communication plan for the consultants;
4. baseline studies, adequacy determinations and tracking procedures, EIS completeness evaluation;
5. budget and schedule tracking and cost controls;
6. goals for environmental enhancement in mine planning and closure; and
7. an informed public affairs process.

20.2.11 Permitting Risks and Risk Management Strategy

This section summarizes certain environmental issues and risks, and strategies by Midas Gold to manage and/or mitigate these risks. The overall approach involves a proactive regulatory/governmental affairs program, which has already been initiated by Midas Gold, and a supplemental environmental baseline program that clearly measures pre-existing conditions at the site. The description that follows highlights those risks; it also lists the measures Midas Gold has put in place to avoid permitting delays and adverse outcomes.

It is possible that the EPA or the Forest Service may initiate CERCLA actions related to historical mining in the Stibnite Mining District and impacts to soils and surface water and ground water quality from these historical operations. Both the EPA and the Forest Service have discussed the possibilities of additional actions at the Cinnabar property, which Midas Gold has optioned for purchase but does not currently own (nor has it conducted any work on the Cinnabar property). A proposed listing for Stibnite on the National Priorities List was rejected by EPA because the risks were determined to be too low to warrant a listing; this determination was made by EPA and the Idaho Department of Health and Welfare, and supported by the Governor, Valley County officials and the Idaho Delegation. However, multiple CERCLA actions not on the National Priorities List have been conducted at previous operations areas:

- Preliminary Assessment/Site Investigation Stibnite Mine Site (USFS, 1993);
- Stibnite Valley Site Inspection, Valley County, Idaho (Greystone, 1993);
- Stibnite Assay Lab Removal Action (EPA, 1998);
- Meadow Creek Diversion (EPA, 1998);
- Stibnite Mine Area Engineering Evaluation/Cost Analysis (MSE, 2003);
- Stibnite Smelter Stack and Tailings Pond Removal South Tailings Pile Contouring Report (MSE, 2003);

- Meadow Creek Channel Realignment (URS, 2005); and
- SMI Mill Removal Action (EPA, 2005).

Much of the previous environmental impact relates to mining activities associated with World War II/Korean War operation of the Yellow Pine Mine (Mitchell, 2000). Between 1939 and 1952, several millions of tons of ore were mined at the Meadow Creek and Yellow Pine mines (see Section 6 for detailed history) and disturbance partially remediated. Other, more recent, mining-related impacts associated with post-1981 projects involving at least four different operators at the West End, Garnet Creek, and Homestake (Yellow Pine area) mines that have also been assessed and partially remediated. In discussions with local and Region 10 EPA officials (as recently as March 2014), the EPA indicated they are not currently considering further CERCLA action(s) at the site. EPA's interest in the Cinnabar property is ongoing.

Midas Gold has met with Region 10 EPA officials to ascertain ongoing interest in CERCLA cleanup at the site. At this time, there appears to be support for a mine plan that incorporates site remediation and fisheries rehabilitation. Midas Gold has also met and confirmed that status with the director of IDEQ. Midas Gold has met with the State of Idaho and local elected officials and the federal Idaho Delegation regarding such a mine plan, as well as discussing the importance of the Project to the local and regional economy (e.g. the Project would potentially represent up to one third of the gross regional product). Midas Gold has also implemented an extensive investigation and documentation of pre-existing environmental conditions at the site. Small individual reclamation and remediation projects have been completed by Midas Gold on an ongoing basis at the Stibnite Gold property. Any potential future CERCLA issues identified from prior operations are considered manageable within the context of the Project.

There is a risk that any single environmental issue or combination thereof, particularly those related to the ESA (i.e. threatened salmon and steelhead species) and the biological opinion, could delay the permitting process. To date, Midas Gold has incorporated specific standard operating procedures (SOPs) and BMPs into its exploration plans to satisfy the requirements to protect these species. An exhaustive biological assessment and environmental assessment prepared for the exploration plan of operations has determined that these activities would have minimal or no effect on the species. These programs would be carried over by Midas Gold to any full-scale mining operation, where appropriate, and others added.

The Project also includes considering a new alternative access route to the site, which would avoid much of the "near waterway" alignment of the existing Johnson Creek and EFSFSR access corridors, reducing the risk of incidents or spills into waterways, or of sedimentation related to roads and traffic thereon.

There is a risk that the NPDES permit for water discharges from the Project would impose stringent water quality criteria. Midas Gold would propose a four-tier system of BMPs, enhanced evaporation standard operating procedures, and contingency water treatment to meet these criteria. The water treatment facilities contemplated in this PFS have been proven at other mining operations located in very sensitive environments.

Risks related to impacts on wetlands, many of which are on top of or impacted by historically impacted mining areas, that are associated with tailings and waste rock placement in the Meadow Creek drainage are planned to be offset by local and off-site wetlands bank acquisition and mitigation projects. The legal authority to place tailings fill in wetlands under appropriate circumstances and criteria is also clear as a result of a 2009 U.S. Supreme Court ruling at the Kensington Mine in southeast Alaska and other precedent.

There is also risk that there may be administration objections to and/or litigation of the outcome of the EIS, or related permitting decisions by the involved agencies.

Midas Gold's risk management strategy focuses on a three-pronged approach. First, the development program highlights the adequacy of the environmental baseline, as discussed earlier in this document. Second, Midas Gold has already established an open dialogue with key environmental organizations, tribal governments, and involved agencies; this has included meetings, site visits, and Project previews with these groups. Third, Midas Gold has proposed a "litigation avoidance initiative" to be formulated with certain key stakeholders that could involve:

1) some level of joint-operational monitoring;
2) input to reclamation planning;
3) employment and business opportunities;
4) third-party environmental audits; and
5) certain other considerations.

The objective is to make the Project a fully integrated, sustainable, and socially and environmentally responsible operation through open communications and accessibility.

The overall permitting process could potentially be expedited by, among other things, negotiating specific permitting agreements or MOUs with involved agencies. The agreements would cover the EIS and/or major permits like the EPA NPDES Permit for water discharge and the USACE 404 Dredge and Fill Permit. These two permits are required for construction of the tailings and waste rock storage facilities, and water discharge from either or both facilities. The MOUs would address:

1) organizational contacts and communication procedures/ limitations;
2) NEPA or permitting objectives;
3) work required to achieve EIS or permit completion;
4) third-party involvement;
5) statement of responsibilities;
6) deliverables;
7) importantly, schedules for review and completion;
8) coordination needs including consultation (Section 106 Historical Consultation and Native Consultation);
9) public involvement requirements; and
10) legal requirements.

20.3 SOCIAL AND COMMUNITY IMPACT

Valley and Adams Counties have experienced continued population growth since 2000, but new job creation is not keeping pace with demand. Much of the downturn is attributed to the construction industry's decrease in the number of jobs. More specifically in Valley County, the Tamarack Planned Community $1.5 billion bankruptcy has had major effects on the local economy. The Stibnite Gold Project would do much to offset these impacts.

Historically, the regional economy has been dependent on wood products. Since 1990, there has been a 43% reduction in jobs associated with logging, wood products, paper production and mining. For Valley County, manufacturing jobs also fell by 67% during the 10-year period of 2000 to 2010. In Adams County, manufacturing jobs fell about 57%. Although slowly declining, the unemployment rates in Valley and Adams County were 10.6% and 12.8%, respectively, in 2013, ranking them 4th and 1st highest in the State. Development of the Project would help revive the mining and manufacturing sectors of the local and regional economy.

According to the 2010 census, the median household income in Valley County was $50,851. The average compensation for a worker was $27,433 which increased in 2012 and then declined again in 2013 (Table 20.3). The average annual salary for a Stibnite Gold Project worker is expected to be consistent with typical Idaho State mining wages of about $72,500 annually (IMA, 2014); this would equate to a total payroll costs of about $88,000 per worker, including benefits. This salary would place Stibnite Gold Project workers among the top five of the occupational categories monitored in the regional economy. Table 20.3 provides current Valley County statistics on average employment and wages for various labor sectors in the county. As seen in the table, in 2012 and 2013 the mining industry had the highest average wage of the 12 labor sectors in Valley County reported on by the Idaho Department of Labor; and mining wages also represent the highest percentage increase (172%) of the 12 sectors over a 10 year period.

Table 20.3: Valley County Covered Employment & Average Annual Wages per Job

Labor Sector	2003 Averages		2012 Averages		2013 Averages	
	Employment	Wages	Employment	Wages	Employment	Wages
Total Covered Wages	3,539	$22,256	3,869	$33,086	3,792	$32,674
Agriculture	77	$18,114	111	$20,121	56	$33,496
Mining	3	$21,929	39	$68,590	35	$81,548
Construction	314	$22,128	269	$33,761	257	$34,166
Manufacturing	48	$20,094	38	$27,536	32	$26,326
Trade, Utilities & Transportation	594	$18,207	649	$28,167	670	$28,806
Information Activities	46	$43,170	167	$69,422	83	$52,963
Financial	128	$21,934	196	$32,815	218	$33,822
Professional and Business Services	179	$27,840	83	$34,256	95	$39,258
Educational and Health Services	186	$23,277	321	$51,453	337	$52,370
Leisure and Hospitality	804	$11,831	936	$17,671	950	$17,884
Other Services	89	$13,937	109	$22,921	102	$18,137
Government	1,071	$31,485	949	$40,232	958	$39,981
(IDL, 2014)						

To estimate the potential economic impacts of the Project, an economic impact model known as IMpact analysis for PLANning (IMPLAN) was constructed to estimate impacts within Valley and Adams Counties (regional impacts), and the state of Idaho. With the exception of federal taxes, the impacts of sourcing workers from outside of the region or the state of Idaho was not assessed, although Midas Gold does expect that a small portion of the workforce might live out of state. These estimates do not represent economic commitments by Midas Gold, but rather the model's simulation of impacts likely to occur if the Project is developed. These estimates may change based on more detailed analysis in future evaluations.

The economic model estimates multipliers for each industrial service sector. Impacts are apportioned into two levels: direct and indirect. Direct impacts are those directly created by the mining operation as export business (i.e. sales of mining products); indirect impacts are comprised of two parts: (1) the impacts on other regional businesses that provide goods or services to mining operations, and (2) the effect of employee and related consumer spending on the economy; these are the indirect and induced effects or impacts, respectively. They collectively comprise the multiplier or ripple effects on the regional and state economy (Peterson, 2014).

20.3.1 Estimated Job Creation and Payroll from Construction of Stibnite Gold Project

During the Project's 3 year construction period the work force would ramp up from an estimated 325 workers directly employed by Midas Gold in year -3 to an estimated 1,000 workers directly employed in year -1. Due to the nature of project construction, workers would be employed from the region and the surrounding areas of Idaho as much as possible; however, to fill the workforce quantity and technical requirements for the Project's construction it is estimated that workers would be sourced from outside of Idaho as well. Estimating the economic impacts and multiplier effects from workers that would work onsite but travel to out of state locations was determined to be too complex to be accurately estimated in the IMPLAN model. As a result the economic impacts generated from the Project for both the construction and operations periods were estimated for the regional economy and the state of Idaho only.

Table 20.4 provides a breakdown of the economic impacts from the construction of the Project complete with annual employment as well as the annual and total compensation paid to workers sourced in the region and the State of Idaho.

Table 20.4: Estimated Construction Period Employment and Payroll

Employment	Annual Personnel	Annual Payroll ($000,000s)	3-Year Total Payroll ($000,000)
Direct (3-yr average)			
Valley & Adams Counties (Regional)	150	$12.5	$37.5
Idaho (Outside of Valley & Adams Counties)	250	$21.5	$64.5
Total Idaho Direct Employment	400	$34.0	$102.0
Indirect and Induced (3-yr average)			
Valley & Adams Counties (Regional)	71	$2.8	$8.4
Idaho (Outside of Valley & Adams Counties)	250	$11.2	$33.6
Total Idaho Indirect/Induced Employment	321	$14.0	$42.0
Total Direct, Indirect, and Induced Employment	721	$48.0	$144.0

20.3.2 Estimated Job Creation and Payroll from Operation of Stibnite Gold Project

As currently planned, Midas Gold would directly employ workers from Valley and Adams Counties and other parts of Idaho over the life of the Project. By comparison it is estimated that jobs created by Midas Gold would be more than double the number of workers employed by the region's largest employer, the USFS (Table 20.5).

Table 20.5: Valley and Adams Counties Large Employers Statistics

Average Employment July 2010 to June 2011 *and Stibnite Gold Project Estimate*	
Business	Employment Range
Estimated Stibnite Gold Project (operations)	*475 to 525*
US Forest Service	200 to 250
McCall-Donnelly School District #421	150 to 200
Valley County	100 to 150
St Luke's Health Systems	100 to 150
Brundage Mountain Co.	50 to 100
City of McCall	50 to 100
Cascade School District #422	50 to 100
Ridley's	50 to 100
Adams County	50 to 100
Evergreen Forest/Tamarack Mill	50 to 100
Council School District #13	50 to 100
Meadows Valley School District #11	25 to 50
J I Morgan Inc.	25 to 50
Adams County Health Center Inc.	25 to 50
The Turning Point	15 to 25
State of Idaho Department of Transportation	15 to 25
(IDL, 2011)	

In addition, the ripple effect from the Project is estimated to provide 120 indirect or induced jobs in Valley and Adams Counties as well as an estimated 329 indirect and induced jobs spilling onto the surrounding areas of the state. In total, the average yearly jobs created during the operational period of the Project are estimated by the IMPLAN model at 939 direct, indirect, and induced jobs.

As noted above, each mine employee would likely generate an additional 0.878 indirect or induced jobs in the Valley / Adams counties economy. Each $1.00 of direct labor income from the hard rock mining industry generates approximately $0.33 of additional indirect/induced labor income. This would amount to approximately $30 million in annual direct and indirect payroll within the region, which is about 10.7% of the region's total gross compensation. Table 20.6 provides a breakdown of the economic impacts from the operation of the Project complete with annual employment as well as the annual and total compensation paid to workers sourced in the region and the state of Idaho.

Table 20.6: Estimated Operating Period Employment

Employment	Annual Personnel	Annual Payroll ($000,000s)	12-Year Total Payroll ($000,000s)
Direct			
Valley & Adams Counties (Regional)	300	$25	$300
Idaho (Outside of Valley & Adams Counties)	200	$17	$204
Total Idaho Direct Employment	500	$42	$504
Indirect and Induced			
Valley & Adams Counties (Regional)	120	$5	$60
Idaho (Outside of Valley & Adams Counties)	329	$9	$108
Total Idaho Indirect/Induced Employment	439	$14	$168
Total Direct, Indirect, and Induced Employment	939	$56	$672

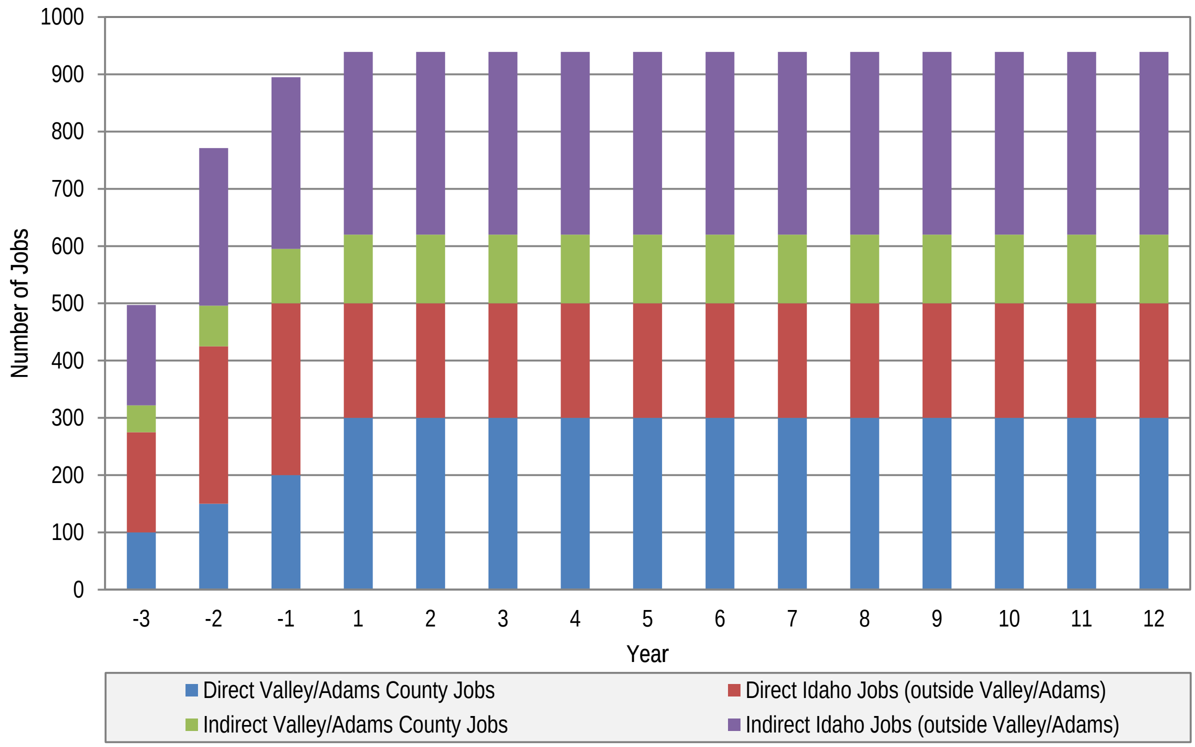
Indirect jobs would include both mine-related and community-based. Key mine-related activities that generate jobs would include:

- trucking of supplies to the mine site;
- access road maintenance;
- trucking of antimony concentrate to the shipment port;
- shuttle buses for the employees;
- catering for the man camp;
- other mechanical and maintenance;
- security services; and
- other supplies services, and contract labor.

Community-based support that generate jobs would include:

- housing construction;
- healthcare;
- law-enforcement;
- restaurants;
- retail shopping;
- public utilities and other infrastructure; and
- schools.

Figure 20.2: Annual Employment Estimates for Construction and Operations



20.3.3 Sales and Taxation of the Stibnite Gold Project

The Project, as currently designed, is estimated to create about $152 million in sales transactions in the regional economy. About $112 million of the sales would be gross regional product, the major subset of sales transactions. Additionally, a large portion of the economic activity created by the Stibnite Gold Project is estimated to affect the rest of the State of Idaho. The State of Idaho sales transactions are estimated to total about $298 million annually (including the regional impacts). The gross state product would total approximately $197 million, providing new economic activity and inter-industry linkages that would help to support every industry in the regional and state economies (Peterson, 2014).

The Project is especially important in Idaho because, according to the Idaho Department of Labor (2012), Idaho consistently ranks nearly last in the U.S. in per capita income and wages. The state ranked in first place in the percentage of workers on minimum wage (IDL 2013). This Project and its high wages will be vital for the region's and the state's economic future (Peterson, 2014).

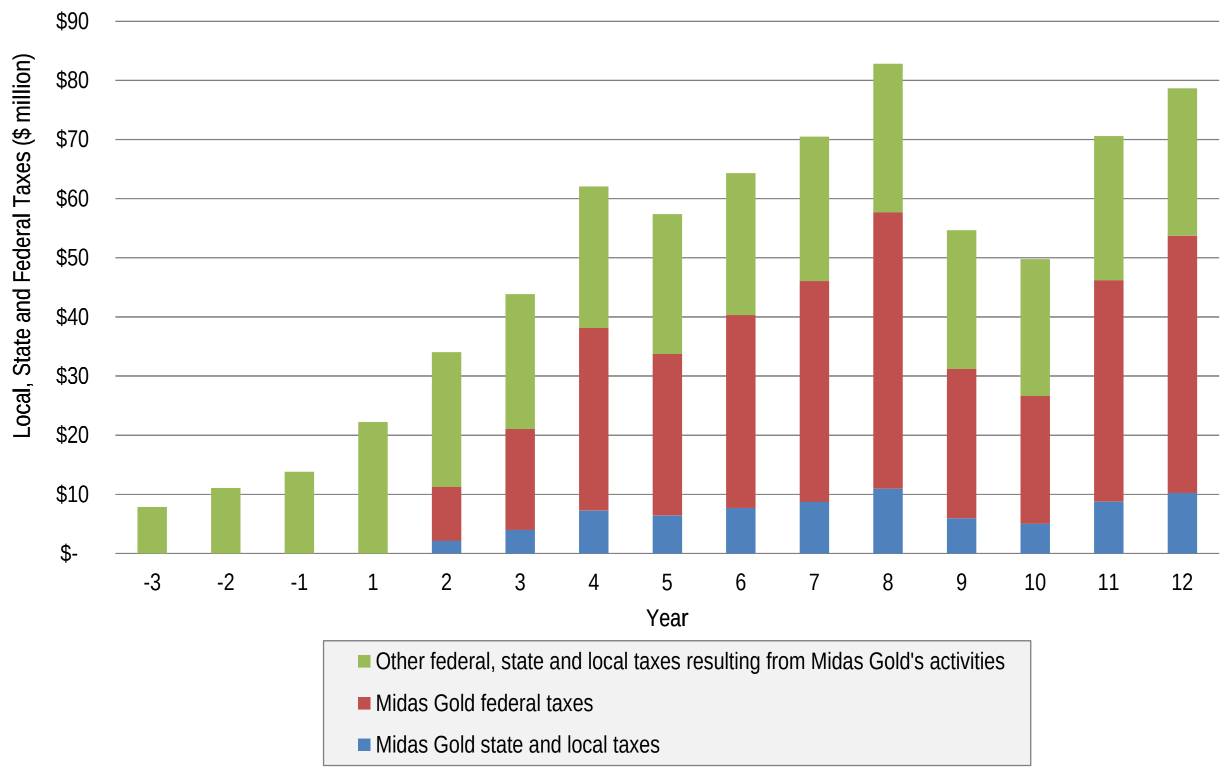
The IMPLAN model was used to estimate direct, indirect and induced taxes, that would be paid by other taxpayers (other than Midas Gold), and the tax estimates were combined with the direct federal, state and local taxes that would be paid by Midas Gold (see Section 22 for details on the PFS financial model and tax calculations) to develop an estimate for the overall taxes generated by the Project. Figure 20.3 presents a plot of estimated annual direct, indirect and induced taxes associated with the Project paid by both Midas Gold and other taxpayers to federal, state and local governments.

Taxes that would be paid directly by Midas Gold over the life of the Project, based on the assumptions in the PFS, are estimated at approximately $329 million in federal corporate income taxes, and $86 million in state corporate income and mine license taxes.

Additional indirect and induced taxes that result from Midas Gold's activities that would be paid by other taxpayers, based on the assumptions in the PFS, are estimated at approximately $177 million in federal taxes (including payroll, excise, income and corporate), and $131 million in state and local taxes (including property, sales, excise, personal, corporate, and other).

Total direct, indirect and induced taxes are therefore estimated at $506 million in federal taxes and $218 million in state and local taxes, representing a significant contribution to the economy during the 15 year construction and operating life of the Project

Figure 20.3: Chart of Estimated State and Federal Taxes



20.4 GEOCHEMICAL CHARACTERIZATION

SRK Consulting was contracted to conduct a geochemical characterization program for Midas Gold as part of the planning and impact assessment for the Stibnite Gold Project. Geochemical testing of mine waste rock materials provides a basis for assessment of the potential for metal leaching and acid rock drainage (ML/ARD), prediction of contact water quality, and evaluation of options for design, construction, and closure of the mine facilities. This work also supports the next phase of the Project's potential advancement, including environmental assessment and permitting. The characterization effort focuses on the assessment of waste rock geochemistry, evaluation of tailings material from mineral beneficiation, evaluation of historic mine waste, and determination of final pit wall geochemistry. In addition, this characterization program includes an evaluation of various historic mine wastes to determine the suitability of reusing these materials as construction materials.

Geochemical characterization is an iterative process and sample collection for the Project is being completed in phases. The first phase is complete and involved the collection of samples from core generated during exploration drilling activities for static and kinetic testing. The characterization program is ongoing and a subsequent phase of sample collection and testing is being conducted to ensure the dataset is spatially representative of the main material types that will be mined as part of the Project.

20.4.1 Waste Rock Characterization

20.4.1.1 Sample Collection and Testing

Waste rock is typically classified and tested according to material type. The number of samples selected for geochemical testing is based on the estimates of the relative percentage of each material type predicted to be mined according to the geologic block model. Material types for the Project were delineated in consultation with Project geologists based on data available from the recent exploration drilling programs, including the drill hole database, drill logs, and assay and multi-element data. During the sample selection process, core holes from the 2009, 2010, and 2011 exploration programs were targeted for sampling and were reviewed in the context of the final pit boundaries using Leapfrog mining software. For the first phase of characterization, a total of 120 sample intervals were selected from within the proposed open pit boundaries to represent the range of waste rock material types that would be encountered during mining and were classified according to rock type and degree of oxidation. Gold grades were also considered and sample selection was generally limited to waste grade samples.

The static test methods used for the mine waste rock characterization program include:

- whole rock analysis using four-acid digest and ICP-MS analysis to determine total chemistry for 48 elements as well as mercury (Chemex Method ME-MS61m);
- Acid Base Accounting (ABA) using the Modified Sobek method (Memorandum No. 96-79) with sulphur speciation;
- Net Acid Generating (NAG) test reporting final NAG pH and final NAG value after a two-stage hydrogen peroxide digest;
- Total Inorganic Carbon (TIC); and
- Nevada Meteoric Water Mobility Procedure (MWMP) (ASTM E2242-02) and analysis of leachate.

These static tests were selected to determine the total acid generation or neutralization potential of the samples and the potential for elemental leaching during meteoric rinsing of freshly-mined material. However, these static tests do not consider the temporal variations that may occur in leachate chemistry as a result of long-term changes in oxidation, dissolution, and desorption reaction rates. In order to address these factors, humidity cell testing (HCT) has been initiated for 14 samples representative of the main waste rock units associated with the Stibnite Gold Project. This testing is currently underway; therefore, the results provided herein are preliminary.

20.4.2 Waste Rock Geochemistry

20.4.2.1 Multi-Element Analysis Results

Multi-element analyses were carried out to provide an absolute upper limit of available metals for leaching from the Stibnite Gold Project material types. These analyses involved the near-complete digestion of a solid sample into solution using a four-acid digest followed by ICP-MS analysis. The multi-element data were analyzed using the geochemical abundance index (GAI) (INAP, 2002), which compares the concentration of an element in a given sample to its average crustal abundance. The results of this comparison show that silver, arsenic, antimony, mercury, sulfur, and selenium are elevated above average crustal concentrations for the majority of the Stibnite Gold Project samples. These elements are typically associated with gold deposits and their enrichment in the waste rock samples reflects the natural mineralization in the area. The actual leachability of these elements was determined from the MWMP and HCT tests that account for factors like the presence of soluble mineral weathering products, mineral habit, rock texture, and site-specific conditions that affect solubility.

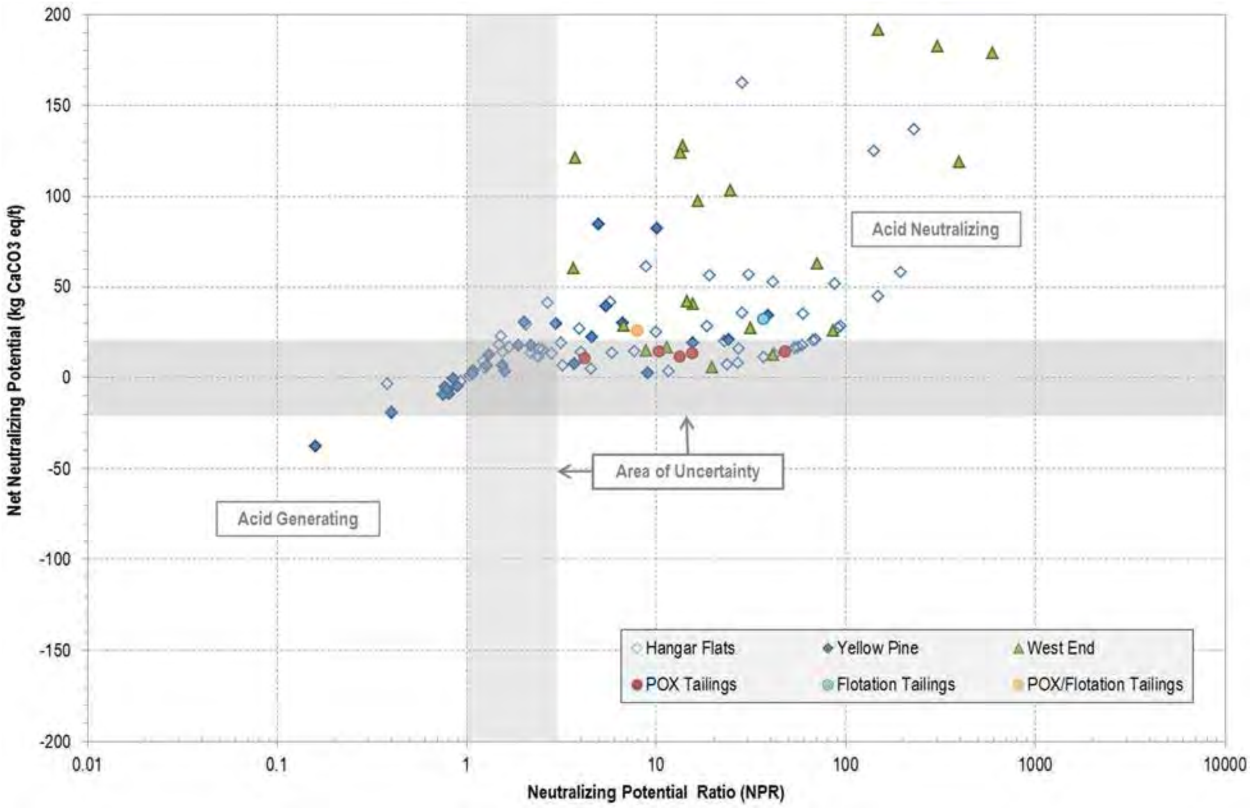
20.4.2.2 Acid Base Accounting Results

ABA tests indicate the theoretical potential for a given material to produce net acid conditions. The balance between the acid generating mineral phases and acid neutralizing mineral phases is referred to as the net neutralization potential (NNP), which is equal to the difference between neutralization potential (NP) and acid potential (AP). The NNP allows for the classification of the samples as potentially acid consuming or acid producing. A negative NNP value indicates there are more acid producing constituents than acid neutralizing constituents. Material that would be considered to have a high potential for acid generation produces an NNP of less than -20 kg $CaCO_3$eq/t. ABA data is also described using the neutralization potential ratio, which is calculated by dividing the NP by the AP (i.e. NP:AP). According to the Nevada BLM Water Resource Data and Analysis Guide for Mining Activities (BLM, 2008), samples with NNP values less than 20 kg $CaCO_3$eq/t and NP:AP values less than three have an uncertain potential for acid generation and require further evaluation (e.g. using kinetic test methods). The NNP values are plotted against the NP:AP values on Figure 20.4. The grey area shown on Figure 20.4 represents a zone in which the acid generation potential of the samples is uncertain.

The Hangar Flats and Yellow Pine samples contain small amounts of potentially acid generating sulfide minerals, with an average sulfide-sulfur concentration of around 0.5 wt%. Despite their overall sulfide content, most of the samples contain neutralization potential in excess of their acid generation potential. Based on the BLM criteria, more than one-third of the Hangar Flats and Yellow Pine samples were characterized as non-acid forming based on BLM criteria, with an excess of acid neutralization capacity; approximately two-thirds of the samples demonstrate an uncertain potential to generate acid and require kinetic testing to determine the long-term potential for these samples to generate acid; only one of the Yellow Pine and Hangar Flats samples shows a greater potential for acid generation with a NNP value less than -20 kg $CaCO_3$ eg/t and NP:AP values less than 1. This sample has the highest sulfide-sulfur concentration (1.5 wt%).

In contrast, samples from the West End Deposit are generally predicted to be net acid neutralizing, with approximately 90% of the West End samples meeting the BLM criteria for a non-acid forming material (NP:AP > 3; NNP > 20 kg $CaCO_3$eq/t); a few samples demonstrate an uncertain potential to generate acid and none of the West End samples show a significant potential for acid generation with NNP values consistently greater than -20 kg $CaCO_3$eg/t and NP:AP values greater than 3.

Figure 20.4: Neutralization Potential Ratio versus Net Neutralization Potential



20.4.2.3 Net Acid Generation Results

NAG testing was carried out in accordance with the method described by Miller et al. (1997). A NAG pH greater than 4.5 s.u. and a NAG value equal to zero are indicative of a non-acid generating material. NAG results greater than one kg H_2SO_4eq/t indicate the sample would generate some acidity in excess of available alkalinity. However, by convention any NAG value below 10 kg H_2SO_4eq/t of material has a limited potential for acid generation.

The NAG results for the Stibnite Gold Project samples are in general agreement with the ABA results and suggest net acid conditions would likely not develop. For the Hangar Flats and Yellow Pine deposits, 85% of the samples are predicted to be non-acid forming on the basis of NAG test work results. From the NAG results, all of the samples from the West End Deposit can be classed as non-acid forming.

20.4.2.4 Meteoric Water Mobility Procedure Results

MWMP tests were conducted on 33 samples to identify the presence of leachable metals and readily soluble salts stored in the material, as well as to provide an indication of their availability for dissolution and mobility. Leachate chemistry data from the MWMP tests were compared to applicable Idaho water quality standards to determine which constituents could potentially be leached at concentrations above these values. However, MWMP leachates only provide a qualitative evaluation of constituents that could occur at concentrations above the water quality standards and do not represent actual predictions of water quality.

The MWMP results indicate the waste rock associated with the Project has a low potential to generate acid or leach metal (loids) from freshly-mined rock. All constituents were below the water quality standards with the exception of trace amounts of antimony and arsenic and, to a lesser extent, aluminum. Antimony and arsenic were consistently leached at concentrations slightly above the water quality standards under circum-neutral conditions, regardless of material type. A comparison of MWMP results from the West End deposit to the Hangar Flats and Yellow Pine deposits, indicate the West End samples show a lower potential for release of arsenic and antimony, which can be attributed to the lower sulfide-sulfur concentrations observed for the West End samples.

The MWMP results are consistent with previous site characterization programs that demonstrate acidic drainage does not occur in the area and arsenic and antimony are elevated in both existing groundwater and surface water throughout the study area. The one exception to this is mercury, which was identified as a constituent of concern from the previous groundwater and surface water investigations, but was below the water quality standards for all samples and below analytical detection limits in all but one sample.

20.4.2.5 Humidity Cell Test Results

Kinetic testing has been initiated on 14 samples to address the uncertainties of the ABA data and provide source term leachate chemistry for the main waste rock units associated with the Stibnite Gold Project that will be used as input chemistry for the predictive geochemical modeling. Sample selection sought to represent the median/mean sulfide-sulfur content for each material type as well as the 95th percentile sulfide content. Samples selected to represent the 95th percentile of sulfide-sulfur concentrations also generally contained arsenic concentrations within the 95th percentile or greater. Likewise, samples selected to represent the median or mean sulphide-sulphur content also represent median or mean arsenic concentrations observed from the static test database.

Samples selected for kinetic testing were submitted for the standard HCT procedure designed to simulate water-rock interactions in order to predict the rate of sulfide mineral oxidation and therefore acid generation and metals mobility (ASTM D-5744-96). At the time of writing, the HCT program is currently ongoing with data available through week 45. The HCT results for the Stibnite Gold Project samples (not including the 3 SODA HCT samples discussed below) show all 14 cells are currently producing moderately alkaline leachates (pH 7.5-8) with low associated sulphate and metals release. Based on the data available to date, constituents are generally below the water quality standards with the exception of trace amounts of antimony and arsenic and to a lesser extent aluminum and manganese. Arsenic and Antimony are consistently leached at low level concentrations above the water quality standards under the moderately alkaline conditions, regardless of material type. These results are consistent with MWMP results and the observed site conditions.

The Stibnite Gold Project HCTs are currently ongoing and will be continued to monitor the potential for development of acid conditions. The HCTs will be terminated when there is no substantial change in the calculated release rate of key parameters or there is no likelihood that changes will occur within a reasonable timeframe.

20.4.3 Implications for Waste Rock Management

The results of the static geochemical test work demonstrate that the bulk of the Project waste rock material is likely to be net neutralizing and presents a low risk for acid generation. However, this prediction needs to be confirmed by the ongoing kinetic testing program since the majority of the Hangar Flats and Yellow Pine samples demonstrate an uncertain potential for acid generation based on the BLM criteria for ABA data. Although the Stibnite Gold Project waste rock material types may present a low risk for acid generation, there is still a potential to leach some constituents under the neutral to alkaline conditions (i.e. arsenic and antimony).

In general, the West End samples show an overall lower potential for acid generation and arsenic and antimony leaching in comparison to the Hangar Flats and Yellow Pine samples. These results suggest effective management of the waste rock could be achieved by using the West End waste rock to cover waste rock from the Hangar Flats and Yellow Pine open pits in order to reduce exposure of waste rock with higher sulfide content to air and water. In addition, management of drainage from all waste rock facilities would be required to limit the release of trace amounts of arsenic and antimony to receiving waters.

For the purpose of the PFS, segregation and selective handling of the waste rock is not considered necessary. However, depending upon the results of the ongoing kinetic testing program and additional sample collection and testing, waste rock from the Hangar Flats and Yellow Pine deposits may require additional management measures such as segregation and selective disposal of potentially acid generating (PAG) material to prevent development of acidic drainage in the long term. In order to demonstrate that the proposed waste management activities for the Project would not result in an environmental impact, numerical predictive calculations would be carried out once the sampling and testing portion of the characterization program is complete.

Characterization of the final open pit wall geochemistry is necessary in order to define the control that the open pit wall rocks would have on the chemistry of waters removed from the pits during operation, and pit lakes that may form after closure. The data from this characterization program is representative of geologic material that would be exposed in the final open pit walls and can be used in subsequent pit water and pit lake modeling efforts for the Project. Pit lake water and pit lake modeling studies will be assessed further in subsequent studies and development by Midas Gold.

20.4.4 Tailings Characterization

20.4.4.1 Sample Collection and Testing

Test residues from metallurgical tests that represent tailings from the Project were sampled and analyzed as part of the current characterization program. The tailings test program includes residues from the bulk flotation tailings, pressure oxidation (POX) circuit tailings, and a mixture of the two. The POX tailings have been run at different grind sizes for the three different deposits providing six samples. In addition, one bulk flotation tailings sample and one sample consisting of an equal mix of POX and bulk flotation tailings have also been characterized, increasing the total to eight samples. Due to the limited quantity of material available, leach testing was completed using a modified Synthetic Precipitation Leachate Procedure (SPLP) (EPA, 1994) in addition to ABA and multi-element analysis.

20.4.4.2 Tailings Geochemistry

From the ABA testwork, the flotation tailings sample included in this study is predicted to be acid neutralizing and contains negligible sulfide-sulfur (0.03 wt%). The sulfide-sulfur content of the POX tailings is comparable to the flotation tailings with an average sulfide-sulfur concentration of 0.04 wt%. However, the neutralization potential for the POX tailings are generally lower than observed for the flotation tailings and, as a result, the acid generation potential for this material is uncertain with NNP values between -20 kg and 20 $CaCO_3$eq/t (Figure 20.4).

From the SPLP results, the potential for metal leaching from the flotation tailings material is demonstrated to be low with the exception of arsenic and antimony that are elevated in low levels above water quality standards under alkaline conditions at 0.35 and 0.025 mg/L, respectively. Therefore, these constituents are predicted to be elevated in the bulk flotation tailings contact water. The source of these constituents is likely from trace amounts of pyrite, arsenopyrite and stibnite remaining in the bulk flotation tailings material.

The POX tailings consist mainly of the oxidation product oxyhydroxy scorodite, a crystalline ferric arsenate mineral and also produced near neutral to alkaline leachates. However, the magnitude of antimony and arsenic release was higher in comparison to the flotation tailings, with an average arsenic concentration of 13.3 mg/L and an average antimony concentration of 0.09 mg/L. In addition, sulfate is elevated above the water quality standards for a few of the SPLP results for POX samples, and weak acid dissociable (WAD) cyanide was above the water quality standards for all POX samples.

20.4.4.3 Implications for Tailings Management

From the test work, bulk flotation tailings are expected to generate neutral pH drainage and require no special disposal considerations to prevent acidic drainage. Although the sulfide-sulfur content of the POX tailings is comparable to the flotation tailings, the neutralization potential of the POX tailings is generally lower and the acid generating potential is currently undefined without further test work. Based on the current mine plan, tailings would be deposited in a single facility, the POX tailings would be co-deposited with the flotation tailings. Due to the excess neutralizing capacity of the bulk flotation tailings, comingling of the POX tailings with the flotation tailings would reduce the overall potential for acid conditions to develop due within the tailings facility. Although acidic conditions are not anticipated to develop, any tailings drain-down would require management to limit the release of trace amounts of arsenic and antimony to receiving waters.

The POX tailings have a lower buffering capacity in comparison to the bulk flotation tailings and the magnitude of antimony and arsenic release is higher in comparison to the flotation tailings. Therefore, there would be some benefit to blending the more reactive POX tailings with the bulk flotation tailings, which is consistent with the PFS approach for tailings storage for the Project.

20.4.5 Spent Ore Characterization

20.4.5.1 Sample Collection and Testing

The characterization program includes an evaluation of the geochemical characteristics of spent ore from the existing SODA. The location and physical properties of this material make it ideal for use as construction material for the Project. A sampling and testing program has been implemented to determine the suitability of the existing spent ore material from the SODA site for use as a construction material. Midas Gold collected material representative of the spent ore material within the SODA site during a comprehensive drilling campaign conducted in 2013. The static test methods used for this characterization program are the same as described for the waste rock characterization program above. Ongoing test work of the SODA samples includes mineralogical analysis and humidity cell testing.

20.4.5.2 Spent Ore Geochemistry

Based on the ABA and NAG test results, the SODA samples contain low levels of potentially acid generating minerals with an average sulfide-sulfur content of 0.07 wt%. Based on the BLM criteria, the majority of the SODA samples are non-acid forming with an excess of acid neutralization capacity. In comparison to the waste rock samples, the SODA samples generally have lower overall sulfide-sulfur and a higher neutralization potential. The higher neutralization potential can be related to the addition of lime to the ore material prior to leaching with cyanide and the source rock for the majority of the material that contributed to SODA being from the West End deposit.

The MWMP results indicate the SODA material has a low potential to generate acid or leach metals with the exception of low levels of arsenic, antimony and WAD cyanide and to a lesser extent aluminum, manganese, mercury, iron, selenium and sulfate. These results are consistent with the results from the waste rock characterization program as well as previous site characterization programs that demonstrate no acidic drainage occurs in the area and arsenic and antimony are elevated in both groundwater and surface water throughout the study area.

Further testing of the SODA material to define the potential for acid development according to the BLM guidance is not warranted. However, three SODA samples were submitted for kinetic testing to assess leaching rates of arsenic and antimony minerals present in the samples. The changes in these reaction rates through the course of the test would be used to estimate the magnitude of constituents that could be mobilized from the SODA material under long-term weathering and oxidation conditions. Samples were selected from the static test dataset to represent the mean sulfide-sulfur /arsenic content, the 95th percentile sulfide-sulfur /arsenic content and the lower sulfide-sulfur/arsenic content (25th percentile). At the time of writing, the HCT program for the SODA samples is currently ongoing with data available through week 16. The HCT results for the SODA samples show all 3 cells are currently producing moderately alkaline leachates (pH 7.5-8.5) with low associated sulfate and metals release. For two of the three samples, all constituents are generally below the water quality standards with the exception of antimony and arsenic. For the remaining sample, aluminum, iron, manganese, mercury, selenium, silver, and sulfate are elevated above water quality standards during the first few weeks of the test and antimony and arsenic are consistently leached at concentrations above the water quality standards under the moderately alkaline conditions.

20.4.5.3 Implications for Use as Construction Material

The results of the static geochemical test work for the SODA samples demonstrate that the spent ore material is net neutralizing and present a low risk for acid generation. Although the SODA material generally presents a low risk for acid generation, there is still a potential to leach some constituents under the neutral to alkaline conditions at concentrations above water quality standards (e.g., arsenic, antimony and WAD cyanide). Based on these preliminary results, the use of spent ore material from the SODA area would be limited to applications where the material would either remain within containment or in a situation where the exposure of the material to air and water is limited. For example, placement of the spent ore below a synthetic liner but above the water table would reduce the potential for further oxidation and mobilization of constituents from this material.

20.5 MITIGATION

In the Stibnite Gold Project design, Midas Gold has created a plan to restore the site by removing existing barriers to fish migration and re-establishing salmon and steelhead fish passage, removing uncontained historic tailings, reusing historic spent ore material for construction, restoring stream channels, and implementing sediment control projects such as repairing the EFMC. In addition to remediating historic disturbance, Midas Gold has minimized the Project's footprint and related impacts by siting facilities and roads on previously disturbed ground and away from riparian areas. Midas Gold has designed the major access route into the Project site to include existing roads for most of its length with some alternate sections designed to avoid rivers and large waterways, resulting in maximum sediment control and related protection of water quality. By including some of the workforce in an office complex in the nearby town of Cascade, the on-site workforce will be minimized, resulting in less traffic. Re-establishment of historic electrical line power to the Project site will serve to minimize fossil fuel consumption and related haulage along the access route.

Midas Gold has also adopted a series of mitigation and conservation principles as follows:

1. Midas Gold would conduct mining, processing, and reclamation activities in an environmentally responsible manner.
2. Project infrastructure would be located on previously disturbed areas and sites where ever practicable.
3. Midas Gold would design and construct facilities to minimize impacts to aquatic and terrestrial wildlife, improve habitat through various projects across the Project site, and protect anadromous and local aquatic populations.
4. Midas Gold would protect and improve local surface water and groundwater quality.
5. Midas Gold would construct or purchase new ecologically diverse wetlands to replace those affected by new mine development.

20.5.1 Mitigation through Responsible Operations

Midas Gold developed and currently utilizes several measures designed to minimize environmental impacts due to current activities at the Project site including:

- A SWPP implemented as part of a Multi Sector General Permit (MSGP) to inhibit sediment or pollution from entering onsite streams.
- A Spill Prevention, Control and Countermeasures Plan (SPCC) comprised of a site-specific spill prevention plan, fuel haul guidelines, fuel unloading procedures, inspections, secondary containment on all fuel storage tanks onsite, and staff training.
- An onsite Recycling SOP to reduce recyclable waste delivery to landfills.
- A surface-water monitoring program to assess the effective implementation of exploration BMPs.
- An annual Environmental Training Program for onsite staff and consultants, covering SWPPP, SPCC, Waste and Recycling management, Midas Gold's wastewater reuse plant, noxious weed overview, Threatened, Endangered, Sensitive, Candidate, and Proposed (TESCP) plants and wildlife overview, and operational requirements.
- An Operating Permit Compliance Training class for site management and supervisors to specifically cover operating permit constraints and limits to promote accountability with all levels of Project management.
- Additional SOPs and BMPs for Fuel Haulage, Drilling, Ground Water Protection, Drill Pad siting and helicopter supported drilling, and reclamation are just some of the various protection measures.

Going forward, Midas Gold would continue to build on their strong record by continuing to proactively evaluate BMPs and SOPs effectiveness, and adapt and improve them as appropriate, including a post-closure component.

20.5.2 Mitigation through Facilities Locations

Careful thought and planning have gone into the Stibnite Gold Project design with specific effort made toward minimizing incremental disturbance by locating facilities and infrastructure on previously disturbed and impacted areas, and improving historic conditions at site, as shown on Figure 20.5. Key examples of this planning effort include:

- The Main WRSF is located at the SODA site, which is also the location of the Historic Tailings storage facility, and has been sited to provide a substantial buttress to the TSF;
- The process plant area encompasses portions of the former Stibnite town site, the current Stibnite camp area, and former contractor shop area;
- The Stibnite Gold Project truck shop and fuel storage area is located on the plant site area from previous heap leach operations;
- The Hangar Flats, West End and Yellow Pine open pits largely lie within areas already extensively disturbed by historical mining operations;
- The EFSFSR diversion approach is similar to that undertaken in prior operations and is situated within the currently disrupted portions of the river channel;
- The Burntlog access road would primarily follow an existing forestry road corridor;
- The power line would follow the existing and historically used power line corridor and right-of-way; and
- Several existing haul road corridors would be utilized to minimize new disturbance.

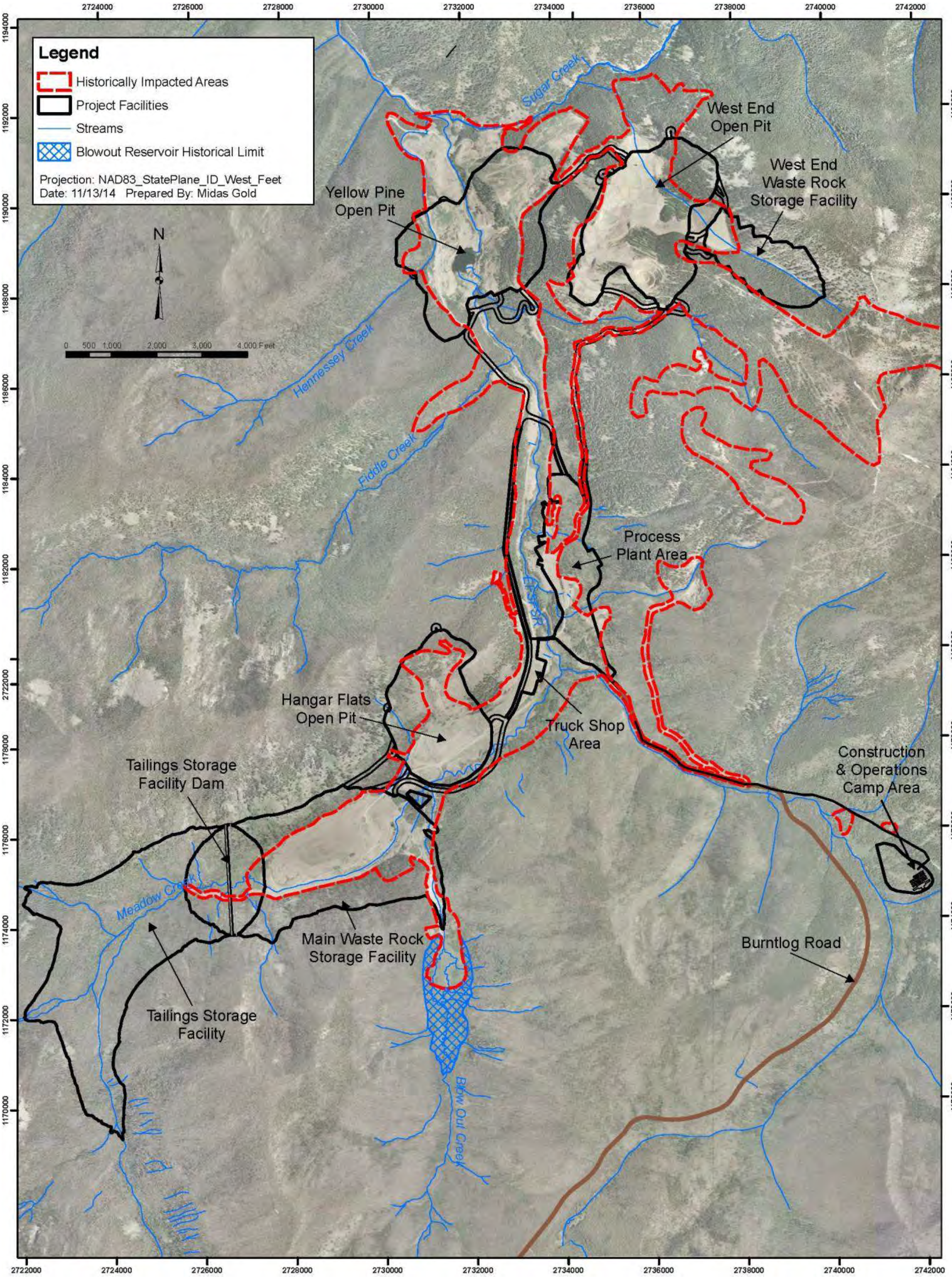
Several examples of improving existing site conditions include;

- Removal of uncontained historic tailings, reprocessing to remove metals in sulfides, and re-deposition of cleaned tailings into a lined TSF;

- Removal and reuse of SODA materials;
- Removal and reuse of spent leach ore from Hecla Mining Company (Hecla) heap;
- Removal of historical waste dumps from around the Yellow Pine and West End pit areas and relocation to a designed WSF;
- Removal of historical Hecla, Canadian Superior Mining Ltd. leach pads and residual infrastructure;
- Removal of potentially contaminated materials from below the historical mill and smelter site;
- Reestablishing short and long-term fish passage through the Yellow Pine pit area on the EFSFSR; and
- Rehabilitation and stabilization of the EFMC.

These are all examples of a Project that has been designed to reclaim the previous disturbance and remnant mining features in order to minimize new facilities and disturbance related to the Project. Major facilities such as the TSF, the Main WRSF, the process plant site, the truck shop, and haul road corridors, have all been sited to minimize total additional disturbance and reduce potential impacts to streams, riparian areas, or wetlands.

Figure 20.5 Project Facilities Locations Relative to Historical Disturbance



20.5.3 Protect Local ESA Listed Fish Species Populations and Enhance Habitat

The protection of fish species and enhancement of local habitat is a key conservation commitment. This commitment involves a "design for closure" approach whereby the development plan, mine plan, and closure strategy integrate key fisheries protection and habitat restoration components aimed at achieving a sustainable anadromous fishery.

In many areas of the Project, opportunities exist during operations and into closure to improve fish habitat by planting trees for shade and installing strategically placed woody species and shade generating plantings. Other efforts that benefit fish include spawning gravel placement, the creation of pools and riffles, and the removal of migration barriers such as the one that currently exists at the Yellow Pine open pit.

Prior to installation of dams on the main Columbia River system and historic mining of the Yellow Pine open pit, the EFSFSR may have been home to a healthy and vibrant salmon spawning run. Since surface mining of the Yellow Pine Pit commenced in 1938, a fish barrier has existed which has halted salmon migration in the District for over 75 years. The Project mine plan, detailed in Section 16 of this Report, entails diverting the EFSFSR and mining first the previously disturbed Yellow Pine pit, followed by mining of the previously disturbed West End Pit, which allows the Yellow Pine Pit to be backfilled with the uneconomic but net neutralizing waste rock portion of the West End Pit. Once the Yellow Pine pit is backfilled with net neutralizing waste material from the West End pit, the grade of the EFSFSR would be restored to approximate pre-mining conditions, thus restoring the in-channel fish migration that existed so many years ago. Upon closure, wetlands and spawning grounds could be established to assist in the return of fish migration and reestablishment of a healthy riparian zone along the rebuilt river channel. As detailed in Section 18 of this Report, one of the main components of the 0.8-mile long EFSFSR diversion tunnel would be a low-flow channel excavated in the tunnel floor, as well as LED lighting, to provide for and encourage passage of migrating salmon, steelhead, and trout to the headwaters of the EFSFSR; this would allow for fish passage during mine operations for the first time since 1938.

20.5.4 Protect and Improve Local Surface Water and Groundwater Quality

Responsible mining operations and a comprehensive monitoring program will serve to protect water quality. Midas Gold may also be able to improve water quality in the District by cleaning up areas that may be impairing water quality conditions. SODA, the Hecla leach facility, waste rock near the Yellow Pine and West End mining areas, and the Historic Tailings facility are previously disturbed areas which may be impacting local ground and surface water and, as a result, have been integrated into the current Project as areas that can benefit through redevelopment and restoration. By relocating these materials to more desirable long-term storage alternatives, their ability to leach constituents of concern should be reduced, such as reprocessing and placement in a lined tailings facility..

Another area for potential improvement is the EFMC, also known as Blowout Creek. This area is a major source of sediment contribution to the EFSFSR drainage basin and results from a water resource dam failure dating back to the 1960s. The EFMC sedimentation could be mitigated during operations when sufficient materials and equipment are available at site. Specifically, one concept would involve installation of a drain system to control sediment production, backfilling of the significant erosional feature, and rerouting of the EFMC stream channel to connect back up with the reconstructed main Meadow Creek channel at mine closure. These efforts would dramatically reduce the sediment currently available for transport at every major precipitation event and during spring runoff.

20.5.5 Enhancement, Restoration, or Creation of Wetlands, Streams, or Habitat

As detailed in this Report, aspects of the current design of the Stibnite Gold Project entail the disturbance of property within the Stibnite Mining District and, in the case of the proposed power-line upgrade and Burntlog Access Road (see Section 18), outside of the District. While Project facilities and infrastructure would be located in areas of previous disturbance wherever practicable, in some cases disturbance of wetlands and streams would be unavoidable. According to current regulations, any person, firm, or agency planning to alter or work in "waters of the

U.S.", including the discharge of dredged or fill material, must first obtain authorization from USACE under Section 404 of the Clean Water Act (CWA; 33 United States Code [U.S.C.] 1344) and, if applicable, Section 10 of the Rivers and Harbors Act of 1899 (33 U.S.C. 403). For work within navigable waters of the U.S., permits, licenses, variances, or similar authorization may also be required by other federal, state, and local statutes. Midas Gold contracted with HDR to conduct an ordinary high water mark (OHWM) analysis and wetland delineation to document baseline wetlands and stream conditions for the Stibnite Gold Project. The delineation was conducted in anticipation of future mine operations that may affect areas containing wetlands and other Waters of the U.S. that may be subject to regulation under the Clean Water Act. The wetland study area encompassed all portions the Project as currently designed. The approximate extent of potential impacts to Waters of the U.S. were determined using HDR's delineation and the mine plans and associated features documented in this Report. The areas that were delineated include portions of the following drainages:

- Meadow Creek;
- EFSFSR;
- Fiddle Creek;
- East Fork Meadow Creek;
- Garnet Creek;
- Midnight Creek;
- Unnamed tributary (commonly known as Hennessy Creek);
- Unnamed tributary (commonly known as Rabbit Creek);
- West End Creek; and
- Sugar Creek.

In order to mitigate for the wetland and stream disturbances, Midas Gold would plan to replace the lost aquatic resource(s) prior to carrying out the planned and approved disturbance. Several means exist to mitigate disturbed aquatic resources, a common one is the use of a mitigation bank. According to the EPA "a mitigation bank is a wetland, stream, or other aquatic resource area that has been restored, established, enhanced, or (in certain circumstances) preserved for the purpose of providing compensation for unavoidable impacts to aquatic resources permitted under Section 404 or a similar state or local wetland regulation" (EPA, 2014). Though it is still early in the Project development, with the current estimated disturbance quantities Midas Gold intends to mitigate Project disturbance through a mitigation bank or similar entity. The potential costs associated with these activities are provided in Section 21 of this report.

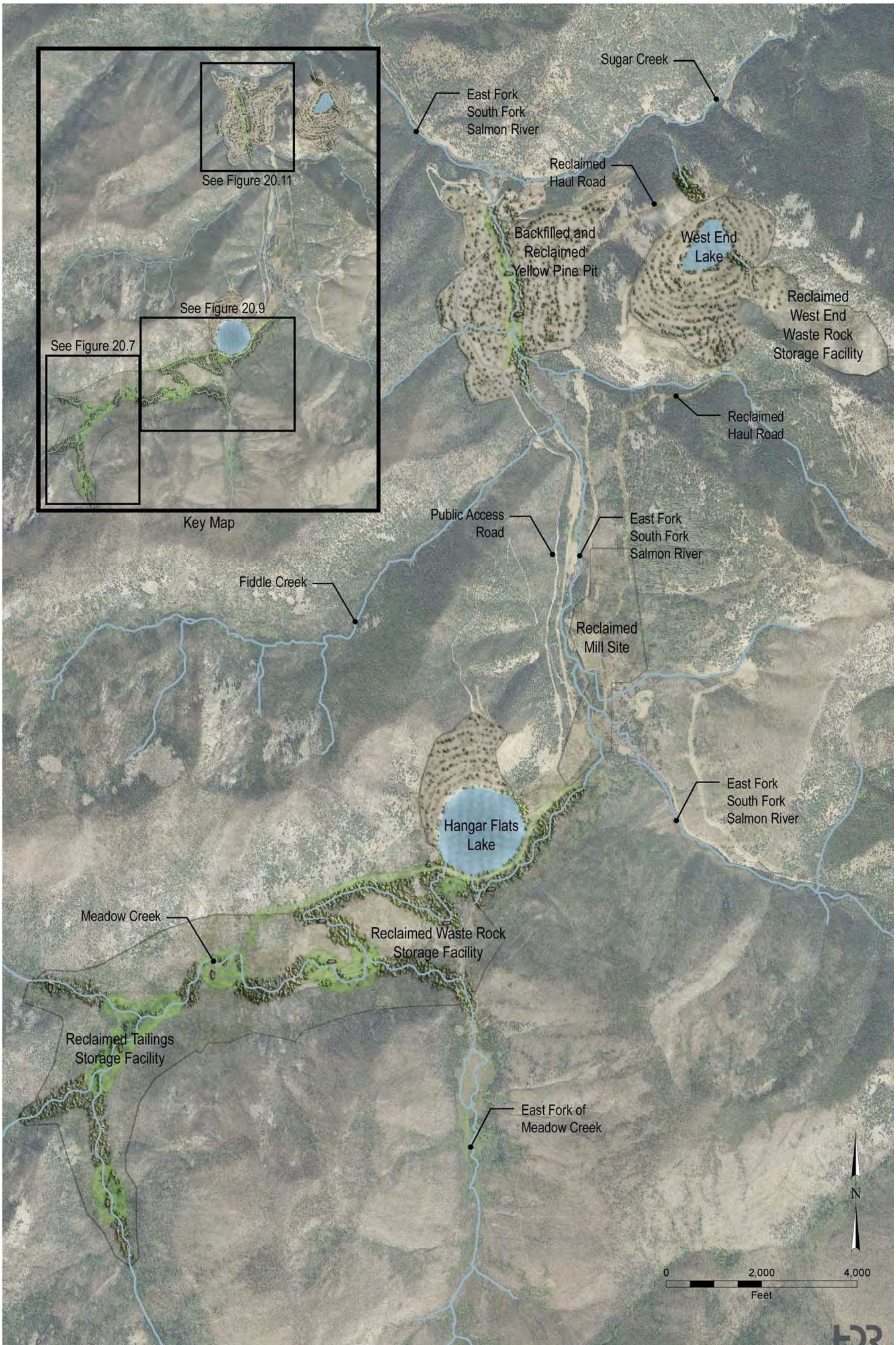
20.6 CLOSURE

Table 20.7 lists 17 priority Project conservation components that form the basis of the overall conservation strategy; Figure 20.6 presents a site-wide illustration of the overall closure strategy. These components range from construction of the new Burntlog Road, which effectively moves the primary transportation route away from the fragile Johnson Creek fishery, to post-closure wetlands and stream habitat enhancement on top of the Meadow Creek TSF surface. The conservation commitment to restore the site through implementation of these measures is also discussed in greater detail, along with associated timing, in the sections that follow.

Table 20.7: Priority Conservation Components

Conservation Measures	Environmental / Conservation Benefit with Approximate Timing
1. Upgrade and utilization of existing Burntlog access route	Elimination of existing access route along high priority fisheries habitats; reduces risk of fuel spills and sedimentation from roads and traffic thereon (Yr -3 to Yr -1).
2. Construct EFSFSR fish passage tunnel	During Yellow Pine pit pre-stripping, construct fish passage tunnel for low/high flow fish passage; allows anadromous and local fish passage (Yr -3 to Yr -1).
3. Construct Meadow Creek diversion around TSF (operational north and south channels)	Divert upper Meadow Creek around initial TSF footprint (Yr -2 to Yr -1) then raise around ultimate TSF footprint (Year 6); removes sedimentation and ground water infiltration from As and Sb source and an existing REC; facilitates effective long-term water management.
4. Spent Ore Disposal Area (SODA)	Relocate 6 Mt SODA facility; provides easily accessible construction material and eliminates significant sedimentation source and existing REC (Yr -1 to Yr 1).
5. Reprocess Historic Tailings	Reprocess 3 Mt of As and Sb laden tailings for permanent storage in a lined TSF; eliminates existing primary source of contamination in Meadow Creek and REC (Yr 1 to Yr 4).
6. Cinnabar access road and Sugar Creek channel rehabilitation	Improve conditions to eliminate major sedimentation source to important salmon spawning reach of Sugar Creek (Yr -3).
7. Relocate Hecla heap leach facility	Relocate Hecla heap leach material to TSF dam; eliminates a potential source of contamination to surface and ground water and REC (Yr -1).
8. Backfill Yellow Pine pit with West End waste rock	Backfill Yellow Pine open pit with geochemically stable material from West End; provides gradient suitable for long-term fish passage in engineered channel and large filter bed of acid consuming material (Yr 7 to Yr 12).
9. Restore segments of Meadow Creek and general habitat below main WRSF	Develop and enhance existing riparian and wetlands and salmon spawning habitat in Meadow Creek downstream of Main WRSF; improves local habitat and removes majority of historic sedimentation and heavy metal contamination from REC source (Yr 6 to Closure).
10. EF Meadow Creek (Blowout Creek) sediment control project	Construct settling basin downstream of Blowout Creek (Year -1) and rock drain, re-contouring and buttressing in Blowout Creek chute (Year 4); enhance settling of turbidity and TSS, minimize major sediment source in EFSFSR basin and re-establish wetlands in valley above.
11. Enhance riparian habitat in EFSFSR above and below Yellow Pine pit	Enhance existing riparian and wetlands and salmon spawning habitat; wetlands mitigation through removal of portions of historic waste rock dumps adjacent to streams and re-establishing habitat in tributary creeks (Yr -3 through Closure).
12. Wetlands and riparian/stream enhancement: Upper Meadow Creek, EFSFSR and off-site purchase	Create and enhance onsite and offsite wetlands according to approved Corps wetlands mitigation and complete closure and reclamation plans (Yr -3 to Closure).
13. Restore Meadow Creek mill and smelter site	Remove material associated with the former mill and smelter site that lies within the limits of the Hangar Flats pit and place contaminated material within the lined TSF (Yr -1).
14. Relocate hazardous waste repository to more suitable location	Relocate the materials in the current hazardous waste repository located on waste dumps adjacent to the EFSFSR to a more suitable location (Yr -1).
15. Closure of historical Bradley tunnel	Closure and sealing of the historical Bradley tunnel, redirecting water into EFSFSR and preventing metal leachates from rock drained by the tunnel entering Sugar Creek.
16. Reforestation of burned and disturbed areas	Extensive tree planting across the entire Project area, which is heavily impacted by forest fires, reducing erosional sedimentation, landslides and avalanches.
17. Closure of historical underground workings on USFS lands	There are a number of historical underground mine workings located on USFS lands that are potential sources of metal leachates and/or safety hazards, including the DMEA portal, that would be sealed reducing or eliminating potential metals leaching and human safety hazards.

Figure 20.6: Overall Site Closure



20.6.1 Tailings and Main Waste Rock Storage Facilities

The reclamation and closure of the TSF consists of two primary elements: a cover system for the tailings deposited in the impoundments, and a surface water management system. The cover is designed to promote vegetative growth and limit erosion on the TSF. The surface water management system (post closure) would direct water across the surface and connect stream segments previously bypassed during operations. By constructing defined channels across the surface of the closed facility, erosional issues would be minimized and in-stream habitat maximized, riparian areas developed, and vegetative growth with native species maximized to reestablish wildlife habitat and stable soil conditions. Flows would be directed into the rehabilitated Meadow Creek stream channel.

TSF closure would begin with removal of the remaining inventory of tailings supernatant on the surface of the TSF through enhanced evaporation, and/or water treatment. Enhanced evaporation using snowmaking misters or similar evaporation systems would limit the need for a new lined pond installation and account for positive water management. Water treatment and discharge would be accomplished under the discharge limits imposed by a NPDES permit, if treatment and discharge are needed.

The mining operation would result in a combined TSF and WRSF complex. The surface of the TSF would be topped at closure with a natural cover to promote vegetative restoration, stream and upland habitat, and control sediment produced from the surface of the facility.

This cover would initially be comprised of sands and gravels and be placed either hydraulically in lenses, or mechanically placed, depending on the composition of the final tailings surface. The cover base (2 - 3 ft) would be amended with soil-like material for planting. The surface would then be re-vegetated to stabilize the facility. Midas Gold would construct a number of vegetative “islands” composed of non-reactive waste rock combined with growth medium across the surface, similar to the existing islands on top of the historic SODA.

Upstream of the TSF, two largely undisturbed stream channels would remain; one to the west, and the other to the south (Figure 20.7). The WRSF is specifically designed to maximize salmon spawning habitat in lower Meadow Creek. To make the rebuilt Meadow Creek channel traversing the WRSF passable to salmon, it would need to be re-sloped thereby reducing the amount of channel in lower Meadow Creek available for spawning habitat, which was deemed a less desirable outcome. The designed slope on the WRSF would potentially allow steelhead, bull trout, and cut throat trout to traverse the facility, thus providing habitat for those species on the upper reaches of Meadow Creek.

The meandering channel on top of the WRSF would offer excellent stable rearing habitat. Habitat engineering could include spawning reaches where properly sized gravel would be located. Some limited periodic maintenance would be required to reposition gravel after extreme high flows.

The channel proceeding down the face of the WRSF would be engineered in a similar fashion as outlined above. The channel would be highly entrenched and installed with a series of pools in a pool and riffle fish-way. Each pool would be keyed into the hardened banks of the channel. The vertical height distance between the successive pools would be a 1 foot to 18 inch transition.

As the channel transitions to the top of the WRSF, its design can revert back to habitat conditions that are reasonably similar to those proposed for lower Meadow Creek. Low flow discharges are common; both depth and wetted channel width decline dramatically at these low flows. A channel 5 ft wide at the bottom that slopes quickly to a maximum depth of 1.5 ft before it widens to benches, would convey higher water depths at any given discharge, than would a wider channel with more gently sloping banks.

The total lined channel would control/contain all high flows without spilling out onto the WRSF, thus avoiding impacts on the impervious layer. Limited high flow associated with the diversion channel and limited sediment inputs by this alignment ensure that the Meadow Creek channel on the WRSF would be very stable. This allows for a wide range of sediment types and habitat features to be installed in the channel. Outside or "off-channel" spawning area comprised of cobbles of 4 to 12 inches, boulders and boulder clusters, or large woody debris, would provide velocity breaks and cover for fish. Riparian plantings of grasses and shrubs, particularly willows, would provide additional cover to the channel.

The need to offset wetland loss associated with the TSF/WRSF complex of fill provides the opportunity to create wetlands on wide benches adjacent to the Meadow Creek stream channel. A construction variation to that described earlier, only with the bench extending a much greater distance from the channel would be employed to accomplish this feature. These "wetland benches" would be planted with a series of succession species, the closest to the channel being the most water dependent.

Figure 20.7 shows a potential succession of plantings from the wettest and most erosive zone to the direct upland and least erosive zone. Plantings from the wettest to the driest are as follows:

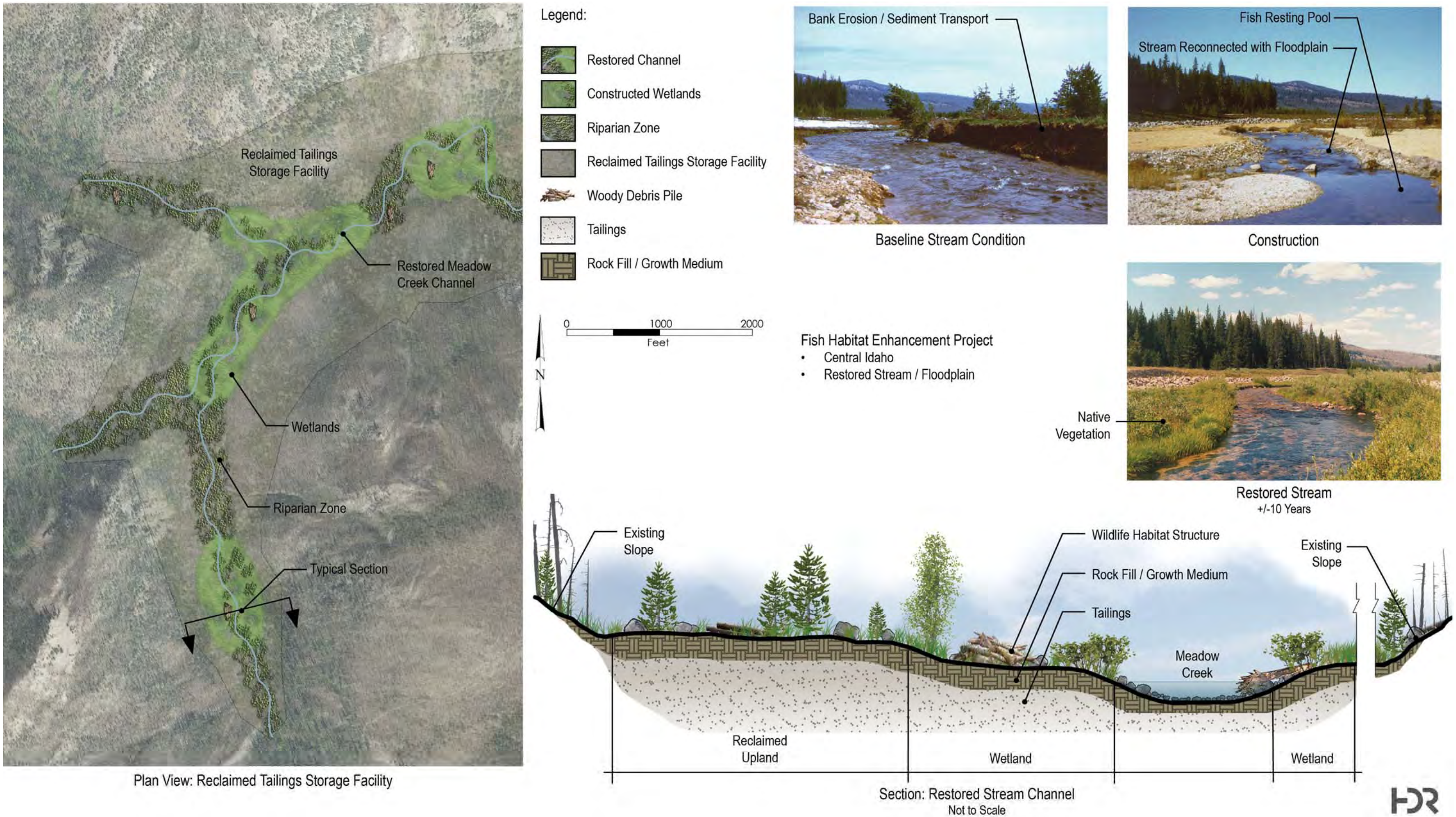
1. Emergent wetland zone;
2. Overbank zone; and
3. Upland wetland zone.

Recommended plant species that are suitable for each zone are described below in Table 20.8. Planting is recommended in the fall. Native stock from the nearby Buffaloberry Nursery or another high elevation nursery is recommended.

Table 20.8: Recommended Potential Planting Zones and Plant Species

Planting Zone	Species Common Name	Species Scientific Name
Emergent Wetland Zone	Nebraska sedge	*Carex nebrascensis*
	Jointed rush	*Juncus articulatis*
	Baltic rush	*Juncus balticus*
	Northwest mannagrass	*Glyceria occidentalis*
Overbank Zone	Coyote's willow	*Salix exigua*
	Yellow willow	*Salix hutea*
	Geyer willow	*Salix geyeriana*
	Black twinberry	*Lonicera involucrata*
	Wood's rose	*Rosa woodsii*
	Aspen	*Populus tremuloides*
	Fremont cottonwood	*Populus fremontii*
	Silver buffaloberry	*Sheperdia argentea*
Upland Zone	Squaw current	*Ribes cereum*
	Common chokecherry	*Prunus virginiana*
	Lodgepole pine1	*Pinus contorta*
	Grand fir1	*Aabies grandis*

Figure 20.7: Tailings Storage Facility Post Closure Details



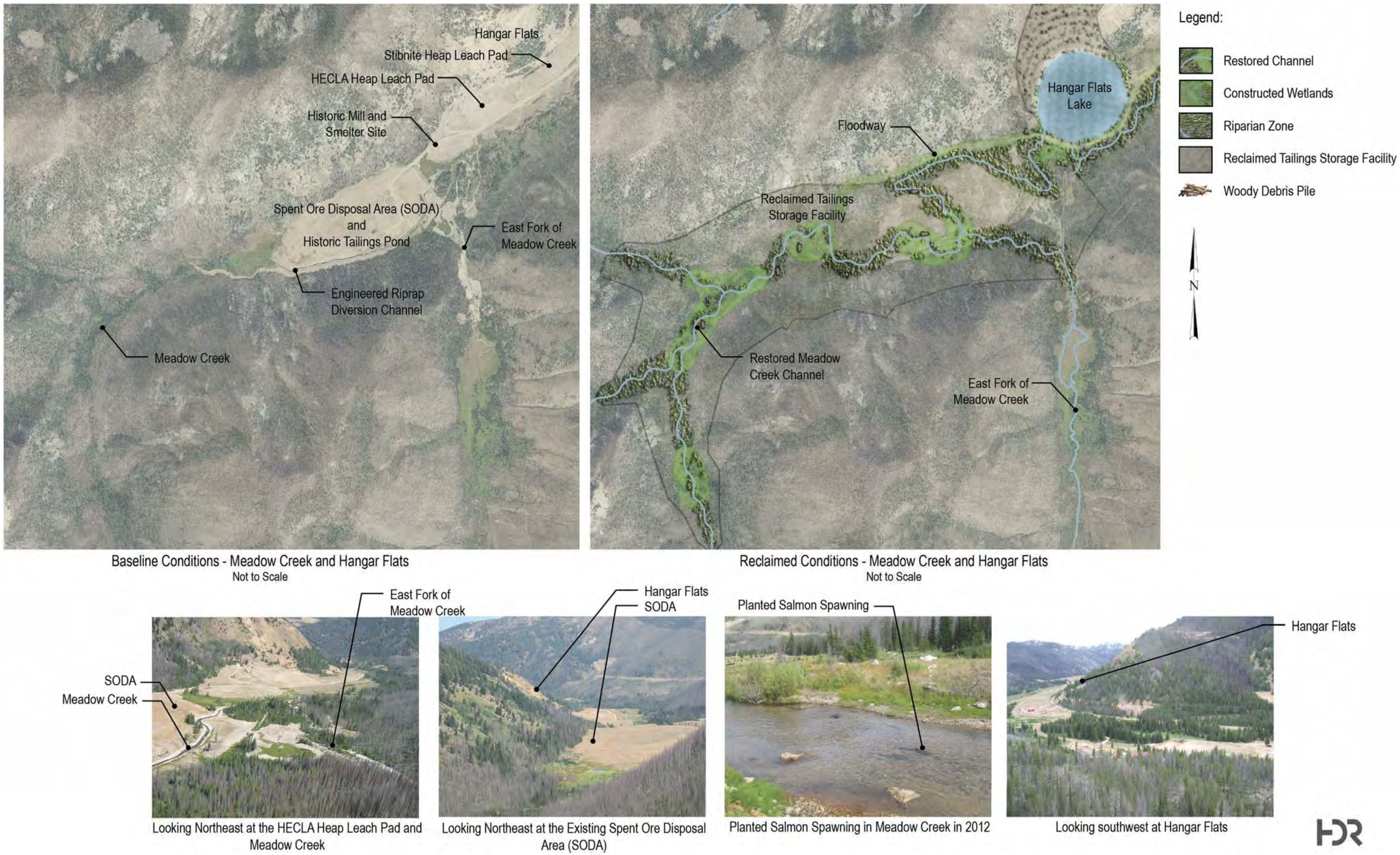
20.6.2 Hangar Flats Open Pit

Hangar Flats pit would function as a sedimentation basin at closure, much like the current Yellow Pine pit does to a reasonable extent today. Surface water and runoff from the TSF, Main WRSF and Blowout Creek would be routed through this "sediment trap" prior to flowing into Meadow Creek. Figure 20.8 illustrates the current configuration of the SODA and Historical Tailings area as well as the Hecla Heap and EFMC erosional features. The pit is projected to be approximately 600 ft deep and 110 acres in size, sufficient volume for hundreds of years of anticipated sediment deposition. It would serve to provide adequate retention needed to settle out a significant part of any suspended solids flowing through the pit. The flow pattern into and out of the open pit is shown on Figure 20.9.

The road leading to the pit would be reclaimed and blocked with large boulders or earthen barriers to prevent motorized vehicle passage. The berms would be placed far enough away from the final pit to prevent failure of the berm due to normal pit wall sloughing. Warning signs will be posted as a safety measure.

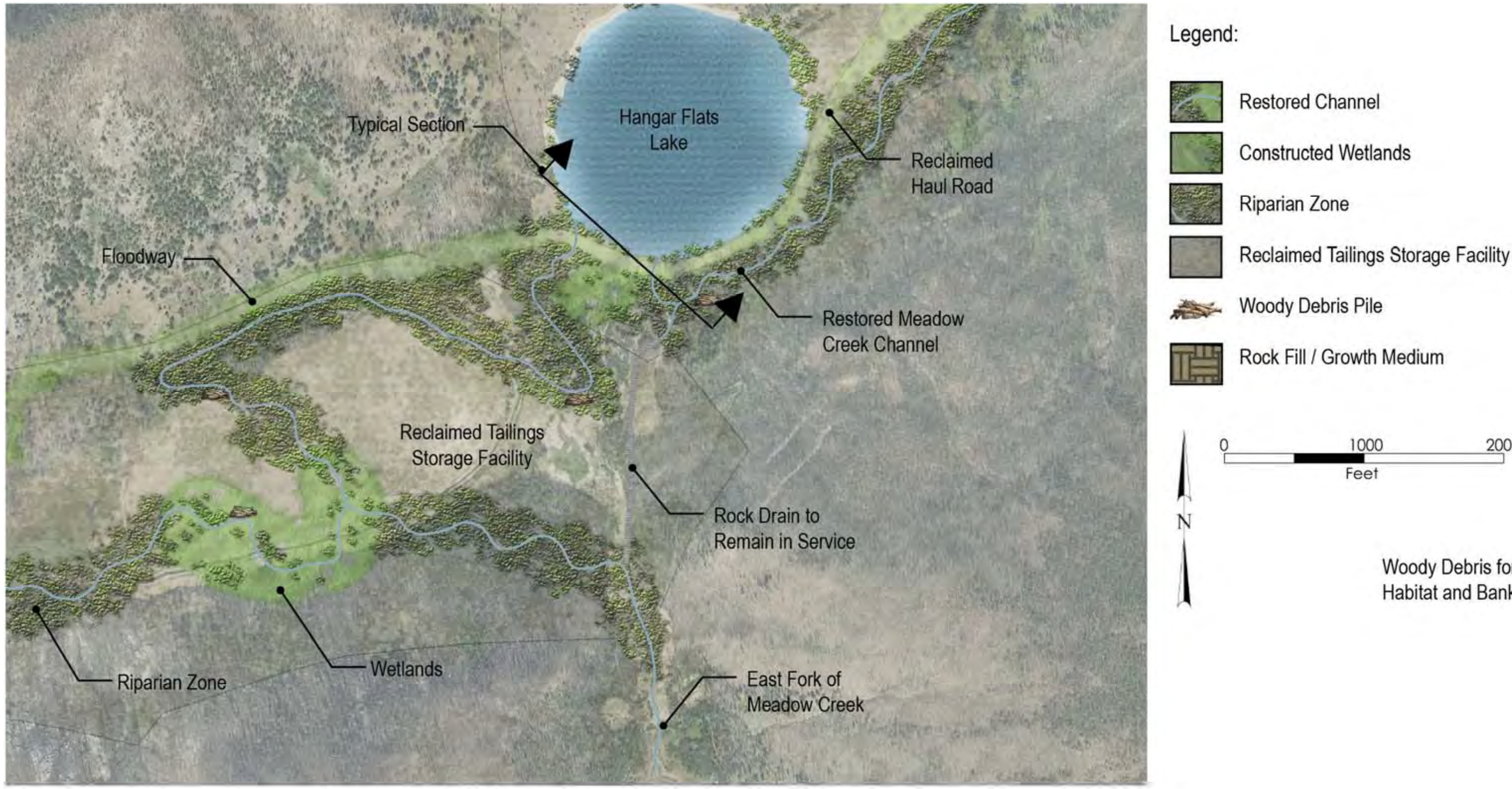
Meander bends would be constructed in reconstructed sections of Meadow Creek all the way to the junction with the EFSFSR. These meanders would be connected by a low gradient straight channel portion of Meadow Creek, which maximizes the length of flowing stream available for spawning, and also reduces the potential for predation of salmon and steelhead smolts by bull trout residing in the Hanger Flats pit.

Figure 20.8: Upper Meadow Creek Area Existing Conditions and Post Closure



HDR

Figure 20.9: Hangar Flats and Main WRSF Area Post Closure Details

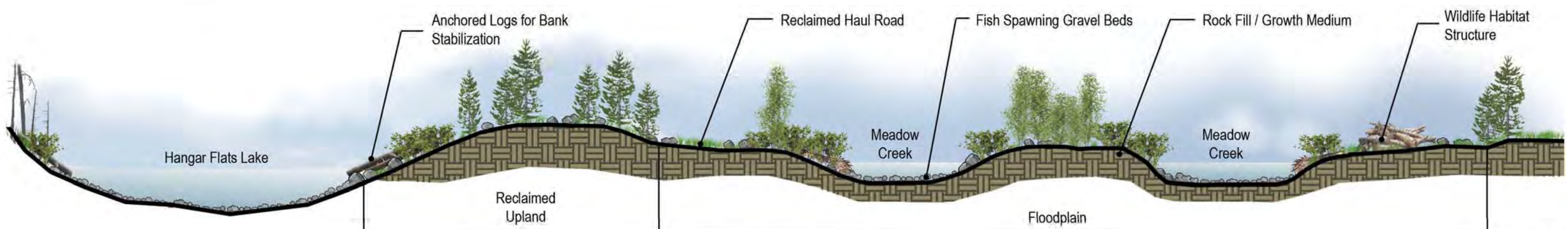


Plan View: Reclaimed Tailings Storage Facility at Hangar Flats

Woody Debris for Aquatic Habitat and Bank Protection

Woody Debris Piles

Wood debris adds habitat for aquatic resources and collects sediment on floodplain areas.



Section: Restored Stream Channel
Not to Scale

HDR

20.6.3 Yellow Pine Open Pit

Initially, the EFSFSR would be diverted around the Yellow Pine pit in a "fish passage tunnel", and reintroduced into the EFSFSR below the pit, and upstream from the confluence of Sugar Creek and the EFSFSR. Based on the anticipated flow regime and fish passage criteria, key considerations are: (1) high flow events; (2) water depths associated with low flow periods; and (3) the vertical height of structures over which salmon and steelhead need to leap while swimming in an upstream direction (water depths and velocities are limiting factor(s)).

Key tunnel design and construction considerations were method of excavation (i.e. single versus dual headings), associated labor, equipment, material costs, low-flow channel configuration, and inlet and outlet channel sections. These components satisfy the following:

- preliminary flood conveyance;
- fish passage;
- maintenance access; and
- pit wall stability-related considerations.

The primary fish-passage design criterion was that fish are able to enter the tunnel, swim the entire length of the tunnel, and exit, continuing their migration without experiencing significant delay or stress. Chinook salmon, steelhead and bull trout, which are protected under the federal Endangered Species Act, are accorded special consideration in the design. Key fish passage considerations include water velocities associated with high flow events, water depths associated with low flow periods, and vertical height structures the fish need to leap over while swimming in an upstream direction.

Upon cessation of all mining activities involving the Yellow Pine pit, Midas Gold plans to restore the stream channel through the backfilled pit by filling and grading a large area of the excavated site. The nature of the backfill material, which is net acid consuming, provides a significant quantity of material that would act as a large scale filter bed to groundwater migrating subsurface down the EFSF valley, potentially absorbing dissolved metals or acidic solutions emanating from upstream, were any to ever develop. A sinuous channel mimicking the EFSFSR channel upstream would be constructed to provide upstream fish passage to trout and salmon spawning and rearing habitat described earlier in the section. The channel will be designed within the step-down benches of the backfilled pit. The service road would be widened in strategic locations to accommodate wetland ponds and riparian habitat. Figure 20.10 illustrates the current configuration of the Yellow Pine Pit area.

High flow events drive the overall channel and floodplain design (width, depth, etc.); a typical channel cross-section would involve an approximate 21 ft wide channel bottom with uniform banks extending to 30 ft wide at bank height, 4 ft above the channel bottom (Figure 20.11) with a substrate of an approximate size of boulders and gravels. The reclaimed channel slope would range from 2.9% to 3.1%, which is comparable to most of the pre-project upstream sections of the EFSFSR. The average gradient of the EFSFSR through the backfilled Yellow Pine open pit is approximately 6% and includes the creation of resting and shelter areas comprised of rock sills for migrating salmon and bull trout.

Once the channel is constructed, the fish passage tunnel would be closed. This would be accomplished by plugging the lower tunnel portal, backfilling portions of the tunnel with rock fill, and plugging the upper entrance. This measure would be implemented late in the overall closure schedule.

Figure 20.10: Yellow Pine Open Pit Existing Conditions and Post Closure

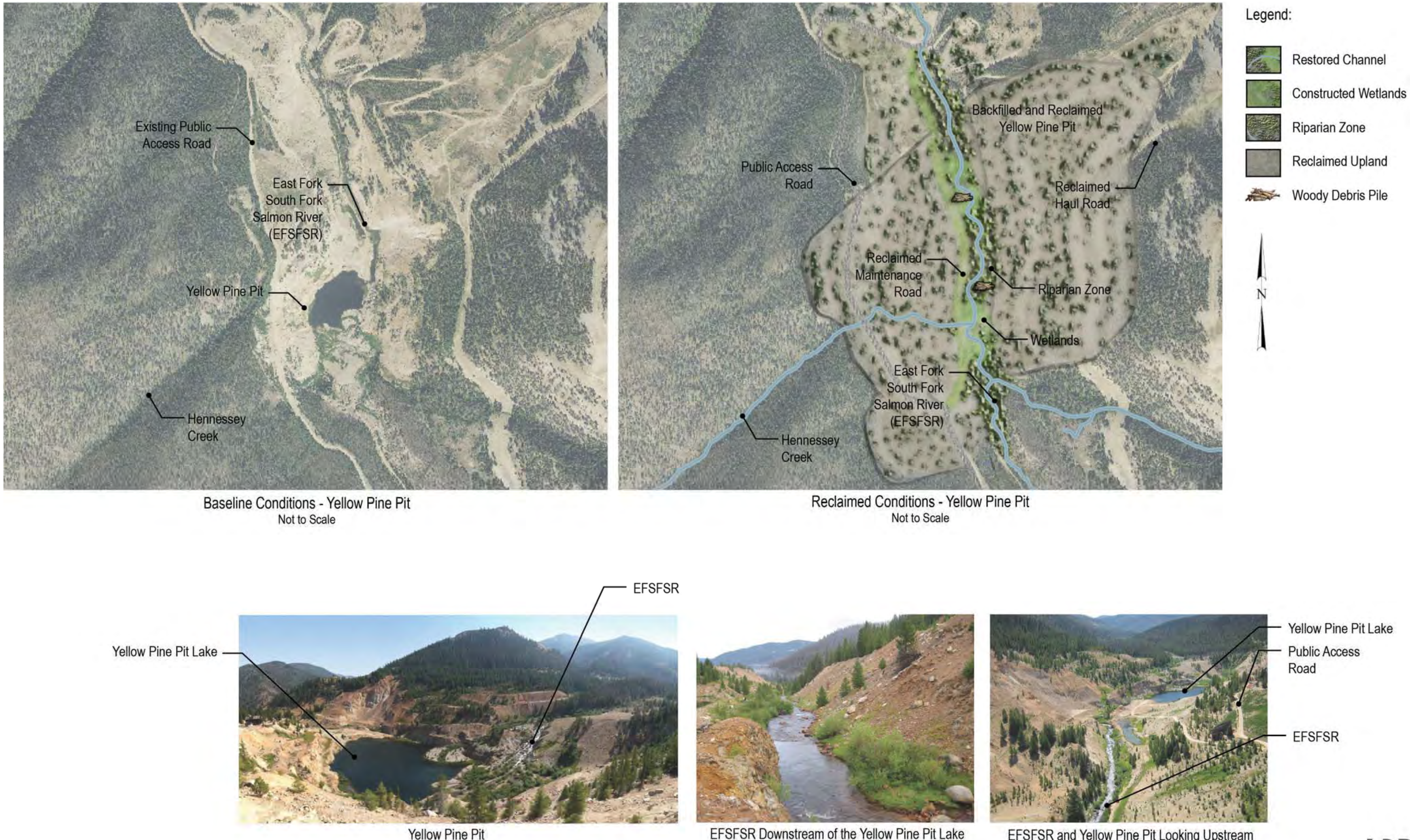
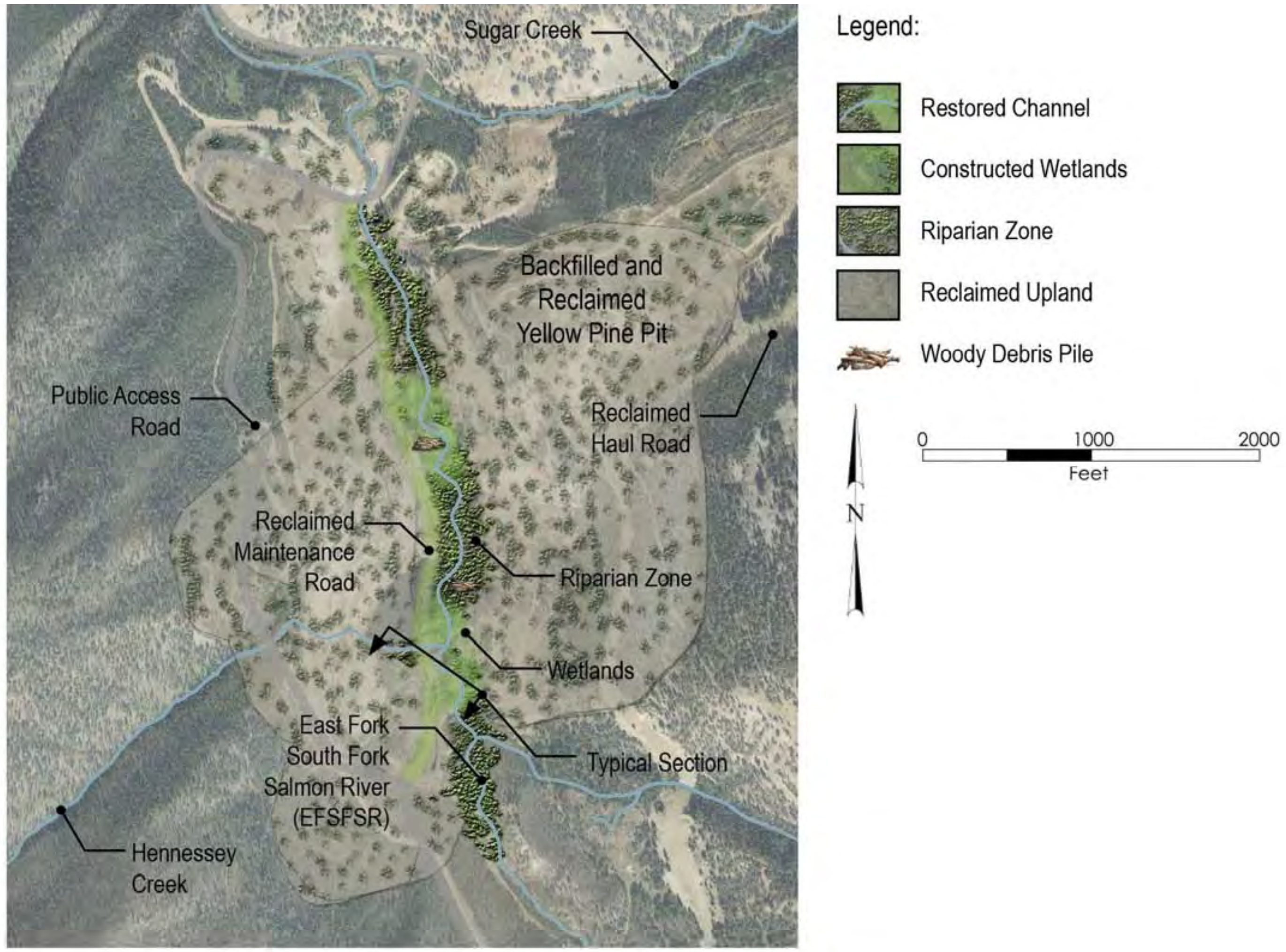
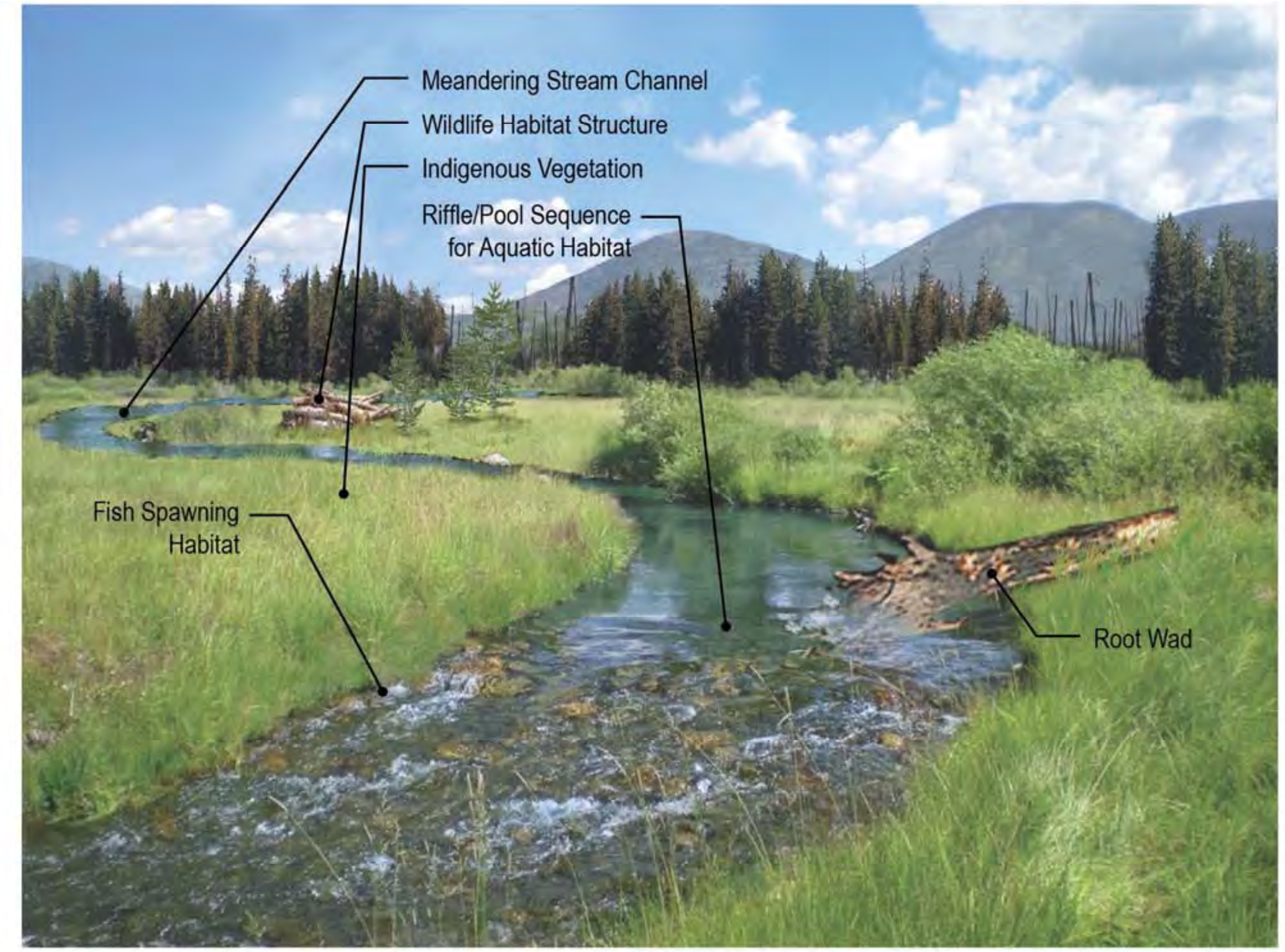


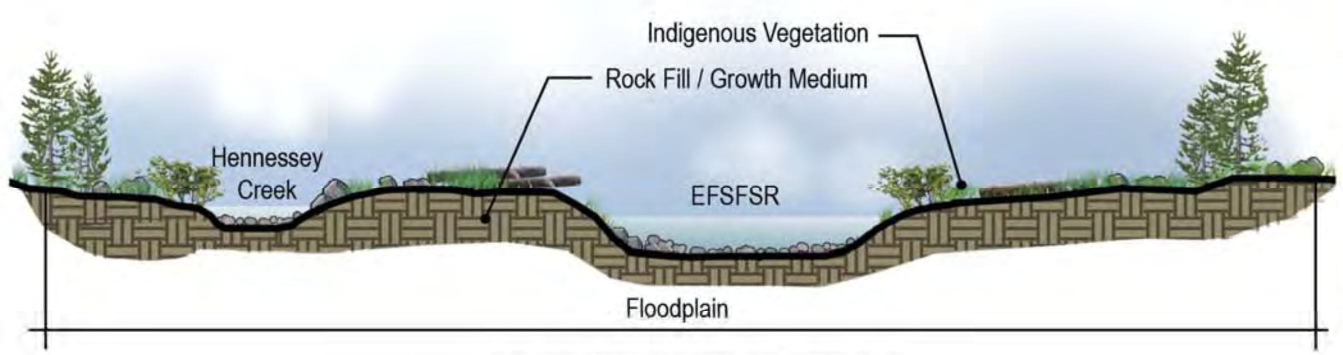
Figure 20.11: Yellow Pine Open Pit Post Closure Details



Plan View: Reclaimed Yellow Pine Pit

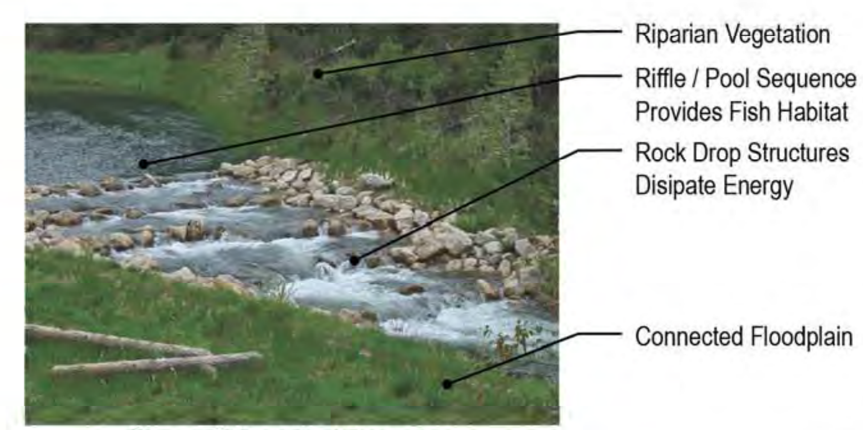


Conceptual Rendering: Reclaimed East Fork South Fork Salmon River Over Yellow Pine Pit Backfill



Section: Restored Stream Channel
Not to Scale

Habitat Structures
Large wood habitat structures maintain pool scour and provide cover for resting fish.



Stream Riffle / Pool Sequence

20.6.4 West End Waste Rock Storage Facility

The West End WRSF would be re-graded to promote positive drainage and prevent pooling on top of the dump; the top of the dump would be crowned and would be covered with growth media to enhance re-vegetation success and limit seepage. The "cover" would then be re-vegetated with an approved native seed mixture to promote stabilization of the facility, and mitigate sedimentation off the surface.

20.6.5 West End Open Pit

Carbonate-rich waste rock from the West End deposit would be used to backfill the Yellow Pine open pit prior to closure. This acid-neutralizing material would form the base over which the new EFSFSR fish passage channel would be constructed. The mined-out West End open pit would encompass an area of approximately 179 acres.

The final open pit closure configuration would be designed so as to not overtop or discharge. This is due to its location high in the drainage with minimal upslope hydrologic catchment. An overflow spillway would channel surface flows along contour to a settling basin, then to lower West End Creek. The West End open pit would be properly signed to communicate the safety risk of the partially backfilled pit lake and pit high walls. Safety berms would also be employed, as appropriate.

20.6.6 Plant Site and Related Infrastructure

Unless there is an ongoing beneficial use, the processing plant, maintenance facilities, office, and shop building will be dismantled and recycled/salvaged to the extent practicable. Foundations will be broken and covered with an appropriate depth of soil-like material (approximately 2 ft).

All non-salvageable equipment would be buried at the site. Fuel tanks would be off-loaded and salvaged. Fuel storage areas would be tested for contamination, as would the areas where the chemical storage buildings were located. All reagents, petroleum products, solvents, and other hazardous or toxic materials in the mill would be removed from the site for reuse, or disposed of according to applicable state and federal requirements.

Following removal of facilities, the area would be graded to promote drainage and re-vegetated with approved seed mix.

20.6.7 Burntlog Access Road

The Burntlog access road upgraded and constructed to avoid or bypass major sections of the Johnson Creek route and its important fishery would be closed once all reclamation work and related environmental monitoring has been completed. Once all reclamation and closure projects are complete, Midas Gold would effectively "un-build" the road; this will involve pulling back and re-contouring new road cuts and fills and seeding with native grasses. Upgraded sections would be returned to their previous pre-project widths and newly constructed road sections would be removed. Valley County and/or the USFS may decide to assume long-term care and maintenance responsibilities for the route. Otherwise, upon removal, access to the site and the neighboring Thunder Mountain area would be re-established by constructing a public access road through the site connect the existing Stibnite and Thunder Mountain Roads.

20.6.8 Operations Camp

The operations camp would be used during the initial 2 - 3 years of reclamation and closure activities but these activities would not require the full camp; consequently, a portion of the camp would be removed during the early years of closure. After the majority of closure activities are complete, the camp would be dismantled and salvaged and the area reclaimed and re-vegetated.

20.6.9 Haul Roads

Strategic roads would initially be left in place during reclamation and closure. Drainage facilities constructed for haul roads would be properly installed including ditches, cross drains, design flow culverts and safety berms. The actual road surfaces and cuts and fill banks would be restored to the pre-hauling configurations by ripping and re-vegetating. Road grades would be designed to facilitate drainage and crowned and fitted with drainage ditches.

20.6.10 Power-Line Corridor

After closure activities that have significant power requirements have been completed, the section of the power-line from the Yellow Pine substation to the site will be disassembled, and the maintenance road reclaimed to its pre-existing state. Drainage stabilization and erosion control features would be installed. The upgraded power-line from Warm Lake to Yellow Pine would be left in place; Idaho Power would continue to maintain that line.

20.6.11 Waste Management Considerations

A number of proactive operational and post-closure water management features would be incorporated into the conservation design strategy. These would include the following:

- There would be construction of a sedimentation basin in the EFSFSR valley bottom and EFMC alluvial fan. These features would re-establish the sediment collection function of the present Yellow Pine open pit lake.
- Later, the EFMC gully would be covered with waste rock to eliminate the source of sediment and the EFMS stream diverted to connect with Meadow Creek along a stable alignment.
- Contact water from the open pits, waste rock, and TSF would be collected and actively or passively treated, or evaporated, or reused in the process.
- Process contact water would be treated to remove contaminants, filter particulates, and control temperature to mitigate potential impacts to local surface water and ground water.
- Midas Gold is also sensitive to the need to maintain adequate stream flows and related in-stream water quality.
- The Meadow Creek TSF is designed to accommodate the Probable Maximum Flood (PMF).

The Hangar Flats pit lake would provide a single point location for future treatment of waters emanating from the TSF, Main WRSF and EFMC (Blowout Creek) should such ever be required. Having the majority of the large scale disturbances feeding into this one location is a significant advantage for such treatment scenarios, if ever required.

20.6.12 LOM and Post-Closure Environmental Monitoring

A conceptual ground water and surface water sampling program follows. A more definitive plan and costs are included in the reclamation plan. Post-closure water quality monitoring would include:

- water quality monitoring as required by the SWPPP;
- water quality monitoring as required by the NPDES Permit;
- ongoing trend sampling for surface water and ground water; and
- monitoring associated with the State of Idaho Groundwater Rule and point(s) of compliance.

The primary purpose of this monitoring would be to determine if potential environmental changes would result from the Project. Further, the monitoring program is intended to evaluate the long-term effectiveness of conservation and mitigation measures outlined in the Final Plan of Operations and other permit approval documents.

Inspections of the TSF and primary waste dump(s) would occur annually for the initial three years following closure, and after extreme events (100-year, 24-hour storm). After this initial monitoring, the contemplated schedule would be Years 5, 15, and 30. This would involve evaluation of the performance of the TSF for the following: geotechnical observations and recommendations, hydrologic monitoring, and water balance review. The actual routine and emergency monitoring and reporting requirements would be defined in the Dam Safety Permit.

The ongoing post-closure fisheries and aquatic biota (stream habitat) monitoring program would focus on evaluating species diversity and habitat conditions as they relate to the Mitigation and Conservation Plan. The initial pre-Project environmental baseline program conducted during 2011-2014 will be the foundation upon which future potential impacts and long-term mitigation success are measured. Key components would include: in-stream flow needs, adult salmon counts, fry escapement and winter survival, habitat characteristics, and construction monitoring. This program would demonstrate conservation and mitigation program effectiveness. Monitoring would occur in Years 5, 15, 30.

Ground water monitoring would focus on measuring any potential changes in exiting ground water conditions beneath the tailings impoundment system and throughout the EFSF basin. Sampling stations below the tailings impoundment, as well as below the three mine pits and the downstream points(s) of compliance, would be indicative of potential ground water impacts associated with the mining operation.

All newly reclaimed areas would be managed consistent with the Project's reclamation, mitigation and conservation principles. The sites would be examined according to the schedule beginning with the concurrent reclamation phase, and proceeding through reclamation and post-closure. The success of re-vegetation would be monitored to ensure erosion is minimized and/or mitigated, and that native species re-establishment is occurring. Maintenance would be conducted on the site as necessary to promote species viability and re-colonization. Reclamation guarantees per 36 CFR Section 228A regulations would be provided by Midas Gold via reclamation bonding or other acceptable and established financial assurance mechanisms.

At the conclusion of "active closure", when construction of all final closure activities is complete, the post-closure program would be initiated. The contemplated schedule is Years 1-5, 15 and 30. Closure maintenance is planned for Years 5, 15 and 30, and would vary for each of the primary components listed.

Midas Gold would compile all reporting information into a single comprehensive "environmental monitoring and mitigation report", based on these schedules. The report would contain information about the following:

- surface water quality;
- ground water quality;
- aquatic biota;
- fisheries;
- tailings storage facility;
- reclamation / re-vegetation status; and
- mitigation and conservation.

The report would be kept on file by Midas Gold, and made available to appropriate federal, state and local agencies upon request.

20.6.13 Conservation and Closure Costs

Anticipated costs for reclamation of the Stibnite Gold Project were developed utilizing a standardized reclamation cost-estimating model currently used and developed in Nevada for mining specific projects. This model has been utilized for mining projects on public and private land in Nevada and other states for many years and is called the SRCE model and is available online through the Nevada Department of Environmental Protection website.

A definitive cost breakdown for reclamation and closure costs and conservation/mitigation measures is provided in Section 21.

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21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

21.1.1 Summary

The estimated capital expenditure or capital costs (CAPEX) for the Stibnite Gold Project consists of four components: (1) the initial CAPEX to design, permit, pre-strip, construct, and commission the mine, plant facilities, ancillary facilities, utilities, and operations camp, and complete on and offsite environmental mitigation and remediation; (2) the sustaining CAPEX for facilities expansions, mining equipment replacements, expected replacements of process equipment and ongoing environmental mitigation activities; (3) the closure and reclamation CAPEX to close and rehabilitate on and off-site components of the Project, which includes post-closure water treatment; and (4) working capital to cover delays in the receipts from sales and payments for accounts payable and financial resources tied up in inventory. Initial and working CAPEX are the two main categories that need to be available to construct a mining project.

Table 21.1 summarizes the initial, sustaining and closure CAPEX for the Project. It includes direct mining equipment and pre-stripping costs, process plant costs, on-site infrastructure such as the TSF and the operations camp, and off-site infrastructure such as the power transmission line, the mine access road, the Cascade Complex, and reclamation and closure costs. The initial CAPEX also includes indirect costs for engineering, permitting, land acquisitions, some environmental mitigation, and other costs. Initial CAPEX also includes an estimate of contingency based on the accuracy and level of detail of the cost estimate. The purpose of the contingency provision is to make allowance for uncertain cost elements which are predicted to occur, but are not included in the cost estimate. These cost elements include uncertainties concerning completeness, accuracy AND CHARACTERISTICS OR NATURE of material takeoffs, accuracy of labor and material rates, accuracy of labor productivity expectations, and accuracy of equipment pricing.

Table 21.1: Capital Cost Summary

Area	Detail	Initial CAPEX ($000s)	Sustaining CAPEX ($000s)	Closure CAPEX ($000s)	Total CAPEX ($000s)
Direct Costs	Mine Costs	47,552(1)	35,346	-	82,898
	Processing Plant	336,219	1,579	-	337,798
	On-Site Infrastructure	149,245	39,937	-	189,182
	Off-Site Infrastructure	80,327	-	-	80,327
Indirect Costs		176,687	4,275	-	180,962
Owner's Costs		26,806	-	-	26,806
Environmental Mitigation Costs		10,606	8,165	-	18,771
Bonding, Closure and Reclamation Costs		762	9,185	56,542	66,489
Total CAPEX without Contingency		828,204	98,488	56,542	983,233
Contingency		142,050	-	-	142,050
Total CAPEX with Contingency		970,254	98,488	56,542	1,125,283

Note:
(1) Initial mining CAPEX also includes some environmental remediation costs as discussed in Section 21.1.7.1.

The primary assumptions used to develop the CAPEX are provided below:

- The estimate is based on 3rd quarter 2014 costs and is un-escalated.
- All cost estimates were developed and are reported in United States of America (US) dollars.

- Contingency during the pre-production period is specific to each major component of the Project as determined by the various consultants.
- Qualified and experienced construction contractors will be available at the time of Project execution.
- Borrow sources are available in Meadow Creek or nearby within the Project boundary.
- Weather related delays in construction are not accounted for in the estimate. However, the engineering, procurement and construction management (EPCM) schedule does account for a ramp down in construction activity during the three winter months (December, January, and February).
- The oxygen plant is accounted for as an “over-the-fence” supply contract. Capital costs have been included for building a dedicated substation for the oxygen plant. Midas Gold will supply power and other utilities to the oxygen plant during operations as well as provide beds at the operations camp for its workers.
- Bonding cost represents the estimated financial cost of putting appropriate bonding in place, not the amount of the bond itself.
- No provision has been made for currency fluctuations.

21.1.2 Mine Capital Costs

The mine capital generally includes three components: capital to purchase the mining fleet, capital for mine support equipment, and the cost of pre-stripping. Mine capital cost for mobile equipment was developed from the mine equipment list presented in Section 16. Mine capital costs including equipment and pre-production development are presented in Table 21.2.

Table 21.2: Mine Capital Cost Summary

Mining CAPEX Components	Pre-Production ($000s)	Sustaining ($000s)	Total CAPEX ($000s)
Mine Major Equipment (Leased)	20,365	27,021	47,386
Mine Support Equipment (Purchased)	16,428	8,325	24,753
Capitalized Preproduction Development (30%)	10,759(1)	-	10,759(1)
Total Mining CAPEX	47,552	35,346	82,898

Note:
(1) Pre-production mining costs include environmental remediation costs as discussed in Section 21.1.7.1; the remaining 70% of preproduction development is included in OPEX as detailed in Table 21.5

Midas Gold plans to lease the major mining equipment, meaning that the majority of the cost of the mining fleet is excluded from initial and sustaining CAPEX and moves to operating expenditures (OPEX). Lease rates were based on a combination of 36-month and 60-month leases mostly with equipment buyouts at the end of the lease period. For some sustaining capital equipment that is planned to be returned to the supplier at the end of the mine life, no buyout was included. Lease rates for the major mine equipment were obtained from a local major mine equipment vendor. End of lease term buyout options are accounted for as capital costs while down payments and monthly payments are treated as operating costs. Starting in Year 1, monthly lease payments become part of operating costs and any buyouts of mobile equipment become included sustaining capital. Smaller support equipment will be purchased, unit costs for which are based on recent information in the IMC equipment cost database. Mine capital costs include the following:

1. Cost of the buyout option for all leased mine mobile equipment required to drill, blast, load, and haul the material from the pit to the appropriate destinations.
2. Auxiliary equipment to maintain the mine and material storage areas in good working order as well as construct the mine haul roads and maintain them.

3. Equipment to maintain the mine fleet such as tire handlers and forklifts.
4. Light vehicles for mine operations and staff personnel.
5. An allowance is included for initial shop tools.
6. An allowance is included for initial spare parts inventory.
7. Mine engineering equipment (computers, survey equipment, etc.) is included.
8. Equipment replacements are included as required based on the useful life of the equipment.

There are certain capital costs associated with the mine that are included elsewhere in the estimate. These items include mine office buildings, shop facilities, mobile equipment that is not required by the mine, and all infrastructure costs (except for haul roads).

Table 21.3 summarizes the mine capital costs by year. The buyout costs for the mine major equipment are included as capital costs, but the down payment and lease payments during the operation period are carried as OPEX and are not shown here. Preproduction stripping is part of the mine capital cost, but is shown separately to differentiate it from the cost of purchasing mine equipment.

Table 21.3: LOM Mining Capital Cost Detail

Year / Quarter	Mine Equipment			Capitalized Preproduction Consumables And Labor ($000s)	Total(1)(2) Mine Capital ($000s)
	Leased Major Equipment Buyout Optional Payments ($000s)	Leased Major Equipment Down & Monthly Payments ($000s)	Other Support Equipment Capital Costs ($000s)		
Initial Capital					
PP Q-5	-	2,221	11,696	857	14,774
PP Q-4	-	5,379	1,942	1,428	8,749
PP Q-3	-	3,307	818	2,545	6,670
PP Q-2	-	3,549	603	2,652	6,804
PP Q-1	-	5,909	1,369	3,278	10,556
Sub-Totals	-	20,365	16,428	10,759	47,552
Sustaining Capital					
1	-	-	39	-	39
2	-	-	53	-	53
3	-	-	30	-	30
4	-	-	933	-	933
5	7,348	-	1	-	7349
6	7,076	-	6,338	-	13414
7	1,821	-	-	-	1821
8	6,365	-	931	-	7296
9	1,494	-	-	-	1494
10	-	-	-	-	-
11	-	-	-	-	-
12	2,917	-	-	-	2917
Sub-Totals	27,021	-	8,325		35,346
Totals	27,021	20,365	24,753	10,759	82,898

Notes:
(1) Mine preproduction development is shown as 30% capital cost and 70% operating expense.
(2) End of lease term buyout options shown as a capital cost

Major mine equipment is leased in the year it is required for operation. The acquisition schedule for the leased major mine mobile equipment is provided in Section 16. The mine capital costs in Table 21.3 represent major mine equipment leased in the preproduction production period; other support equipment is purchased outright. Equipment leases after the start of production are carried as OPEX.

If major mine equipment was purchased outright, the initial CAPEX would increase by $53.4 million, sustaining CAPEX would increase by $53.7 million, and LOM undiscounted OPEX would decrease by $121.8 million.

Mine support equipment will be purchased outright. Table 21.3 also includes the mine support equipment capital costs. Mine support equipment pricing was priced from vendor quotes and from IMC's database for capital equipment purchases. The truck shop, truck wash, and truck shop warehouse are included in the Plant CAPEX. Table 21.4 presents the detailed purchase schedule for the support mine equipment along with the necessary facilities capital costs.

Table 21.4: Mine Support Capital Equipment and Facilities Capital Costs

Mine Support Equipment / Facilities	Unit Cost ($000s)	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Blasthole Stemmer	317	317	634	951
Blasters Flatbed Truck (2 T)	80	160	160	320
ANFO/Slurry Truck (40,000 lb)	511	511	511	1,022
Fuel Truck 777G	1,511	1,511	1,511	3,022
Lube Truck 777G	1,678	1,678	1,678	3,356
Flatbed Truck (8 - 10 ton)	102	102	102	204
18 T Service Truck with Crane	316	316	316	632
Crane Truck (8 - 10 ton)	296	296	296	592
Cat 988 with Tire Handler	682	682	682	1,364
Mechanics Truck	166	498	498	996
Welding Truck	277	277	277	554
Shop Forklift (Hyster H100FT)	52	52	-	52
RT Forklift (Sellick SD-100)	90	90	-	90
Grove 80 T Crane	488	488	-	488
Cat 430E Backhoe/Loader	157	157	157	314
Man Van	71	284	568	852
Pickup Truck (4x4)	66	330	660	990
Light Plants	30	150	90	240
Mine Radios	1	35	35	70
Mine Dispatch System	2,650	2,650	-	2,650
Engineering/Geology Equipment	150	150	150	300
Shop Tools (3% of Major Equipment)	2,135	2,135	-	2,135
Initial Spare Parts (5% of Major Equipment)	3,559	3,559	-	3,559
Total Support Equipment/Facilities CAPEX		16,428	8,325	24,753

Pre-stripping requirements were developed quarterly to provide ore exposure for production in year 1 and also construction material for the TSF starter dam. A total of 14.4 Mst of waste rock would be mined during preproduction. Approximately 14.1 Mst (5.5 Mst from YP, 2.3 Mst from HF, 0.5 Mst from WE, and 5.8 Mst from SODA) of the 14.4 Mst would be used to build the TSF starter dam; the remaining 0.3 Mst of waste rock from WE would be stored in the WE WRSF. Mining costs during pre-production were based on areas stripped, haul profiles, established equipment rates and estimated operator wages. The cost build-up assumes that pre-stripping activities will be conducted by an owner-operated fleet using leased equipment. Table 21.5 shows the estimated development costs by quarter before start-up. The costs for topsoil stripping and storage are included in the mining costs. Development costs in preproduction quarter -5 (PP Q-5) include costs for the haul road between the Yellow Pine pit and the WRSF and TSF. The development costs are divided between capital (30%) and operating (70%) expenses.

A small amount of capital is needed to support the mining effort, as shown at the bottom of Table 21.4. These include geology and engineering equipment such as surveying instruments, and mine planning hardware and software. Truck shop initial tools were estimated as 3% of mine major equipment capital costs. The first supply of the truck shop warehouse with operating spare parts is capitalized at 5% of mine major equipment capital costs.

Table 21.5: Mine Pre-Production Expense

Period	Development Costs ($000s)	CAPEX 30% ($000s)	OPEX 70% ($000s)
PP Q-5	2,856(1)	857	1,999
PP Q-4	4,760	1,428	3,332
PP Q-3	8,482	2,545	5,937
PP Q-2	8,840	2,652	6,188
PP Q-1	10,925	3,278	7,648
Total	35,863	10,759(2)	25,104(2)

Notes:
(1) Dedicated for haul road development.
(2) Mining CAPEX and OPEX include environmental remediation costs as discussed in Section 21.1.7.1.

21.1.3 Plant Capital Costs

Capital costs for the processing plant were estimated using budgetary equipment quotes, material take-offs for concrete, steel, and earthwork, estimates from vendors and consultants, and estimates based on experience with similar projects of this type. The capital cost estimate for the plant is shown in Table 21.6. Some of the costs and quantity estimates used by M3 were supplied by other consultants.

Table 21.6: Plant Capital Cost Summary

Area Description	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Historic Tailings	4,565	-	4,565
Primary Crusher	12,108	-	12,108
Crushed Ore Stockpile & Reclaim	8,884	-	8,884
Grinding and Classification	61,716	-	61,716
Pebble Crushing Circuit	-	1,579	1,579
Antimony Recovery	22,346	-	22,346
Gold Flotation	25,733	-	25,733
Pressure Oxidation	71,934	-	71,934
CCD & Neutralization	39,254	-	39,254
Leaching/CIP	17,346	-	17,346
Detox	2,943	-	2,943
CIL Leaching	14,535	-	14,535
Carbon Handling & Refinery	10,367	-	10,367
Fresh Water System	2,336	-	2,336
Main Substation	20,722	-	20,722
Reagents	18,214	-	18,214
Oxygen Plant	3,217	-	3,217
Total Plant CAPEX	336,219	1,579	337,798

21.1.3.1 Plant Capital Basis of Estimate

The capital cost estimate is based on the cost of equipment, material, labor, and construction equipment needed to complete the plant up to start-up. The accuracy of the CAPEX estimate at the prefeasibility level is -10% to +25. Data for this estimate was obtained from numerous sources including:

- prefeasibility-level design engineering consisting of flow sheets, general arrangement plans and cross sections, civil grading drawings, and electrical one-line drawings;
- POX conceptual engineering produced by Pieterse Consulting Inc. during 2014;
- topographical base information provided by Midas Gold from a 2009 aerial LiDAR survey augmented by a 2013 LiDAR survey for outlying areas for the mine access road;
- budgetary equipment and materials quotes from vendors; and
- local labor rates for Valley County, Idaho modified by M3's experience from other projects.

Below is a description of the pricing that was used by category.

<u>Capital Equipment Pricing</u>

Prices were solicited for all major equipment. Procurement packages of similar equipment were sent to three qualified suppliers to get budgetary quotations. Major capital equipment categories for this Project included electrical, mechanical, and piping. Accuracy of +/- 15% was requested from suppliers for this CAPEX. For some equipment generally under $100,000 in value, pricing data were taken from recent M3 projects.

Electrical Equipment

One-line electrical distribution diagrams were designed for each plant and ancillary area to determine the required number and size of transformers, switchgear, and motor control centers. These one-line drawings were sent to three qualified electrical suppliers for direct pricing. Quotes were evaluated by the electrical engineer to ensure that the specifications for the equipment were met. An average of the suppliers quoted prices was used to populate the capital cost estimate.

Electrical bulk materials were factored by area and benchmarked from recent projects. The cost of electrical equipment was subtracted from the factors except in cases where the electrical costs were judged to be too low.

Mechanical Equipment

All major mechanical equipment was priced for the capital cost estimate by soliciting budgetary quotations, or in the case of minor equipment, from quotes or purchases from recent jobs. The vendors that were approached were generally the best known suppliers of process equipment in the mining industry: Metso, FL Smidth, Outotec, Sandvik, Weir Warman, GIW, Goulds, Flowserve, Delkor Tennova, McClanahan, Konecranes, etc. Autoclave equipment prices were solicited from the main providers: Koch-Knight, Stebbins, DSB, Ekato, Lightnin, Mikropul, Clean Gas Systems, Weir-Geho, Midwest Cooling Towers, Marley, Clayton, Cleaver-Brooks, and others. Operating data sheets (ODSs) were developed to provide duty specifications for each unique piece of major equipment in the Equipment Register. The ODSs were populated with process data and flows from the Metsim process simulation, from specifications in the Process Design Criteria and from physical information derived from General Arrangement drawings. Vendors were provided capacities and flows (nominal and design), specific gravity and bulk density, slurry densities (percent solids), work and abrasion indices, materials of construction, and other information needed to receive a credible quote. All quotes were evaluated to determine if they met the duty specifications. In general, the price that was used in the capital cost estimate was based on the best suited proposal, not the lowest cost quote. In cases where there were multiple qualified proposals, the average of vendor's prices was used. Mechanical equipment quotes were obtained for:

- jaw crusher;
- conveyors & stacker;
- reclaim apron feeders;
- SAG/ball mills with trammel screens;
- pebble (cone) crusher;
- hydro-cyclone;
- flotation tank cells for both antimony and gold rougher and cleaner circuits;
- concentrate, CCD, and neutralization thickeners;
- plate and frame filter presses;
- field erected and shop-fabricated tankage;
- POX equipment including the autoclave, flash tanks, autoclave agitators, positive displacement feed pumps, steam generators, and Venturi scrubber;
- conventional 7-ton carbon plant for gold recovery and carbon regeneration;
- screen slurry plant for Historic Tailings; and
- tailings, slurry, froth, and process pumps.

Quotes for the autoclave, flash tanks, and autoclave agitators were based on preliminary sizing information. The final process design required a larger autoclave and flash tanks. Prices for the final design were factored to reflect the new autoclave size. For the autoclave and flash tanks, a factor of 21% represents the estimated increase in steel weight of the larger autoclave over the autoclave that was quoted. Autoclave agitators, positive displacement feed pumps, and other scalable equipment were scaled up similarly.

Piping, Pump, and Valve Quotes

A list of pumps was developed for all process areas. Operating data were tabulated for all pumps on this list including flow, total dynamic head, percent solids, slurry specific gravity, service, corrosivity, and pump style (horizontal centrifugal, vertical turbine, etc.). Requests for budgetary quotes were furnished to three or more pump suppliers for comparative quotes. A piping engineer reviewed the vendor submissions and technical information to select the appropriate equipment to include in the capital cost estimate.

M3 sized and specified the valves in the autoclave area. These valves were priced by known providers, Caldera, Mogas, Flowserve, and Tyco. The total bill of materials for autoclave area valves is $6.5 million. Pipe costs were factored and benchmarked against other recent projects.

Structural Steel and Concrete Material Take Off (MTO) Methods

The method used to quantify concrete volumes and structural steel weights was to take similar structures already designed from other projects and then modify the dimensions and/or steel member sizes appropriately for the new project's building characteristics. At times when there are different loading conditions, (e.g. snow load), a quick calculation was performed to verify sizing of roof framing. When there was a new foundation that was judged to be unique from past projects, a calculation of a typical bearing condition was performed to verify concrete dimensions. The calculations are effective at getting a close approximation of the final design.

Concrete & Structural Commodity Pricing

MTOs were prepared for architectural and structural commodities to establish quantities and prices using solicited budgetary prices of unit costs. Unit pricing was solicited from four structural steel providers for the Project, which were adjusted for steel unit prices typical for current large EPCM jobs.

Two regional concrete suppliers and contractors provided prices for supply of concrete, predicated on the assumption that a batch plant would be set up on site and that aggregate would be available from alluvial gravels in the Meadow Creek valley. A crushing and screening plant would also be needed to make the particle size gradations for concrete mix designs. The cost to house the batch plant operators was also considered.

Plate work was estimated using MTOs at a cost of $2,850 per ton.

Instrumentation

Instrumentation materials costs were factored by area from total plant equipment costs.

21.1.4 Infrastructure Costs

21.1.4.1 Onsite Infrastructure

The onsite Infrastructure includes site utilities and roads, auxiliary facilities, the TSF, surface and tunnel water diversions, and the operations camp. Table 21.7 summarizes the direct costs for onsite infrastructure.

Table 21.7: Onsite Infrastructure CAPEX Summary

Onsite Infrastructure	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Site - General	21,521	-	21,521
Auxiliary Facilities	38,552	-	38,552
Tailings Storage Facility / Reclaim System	42,757	25,371	68,128
Water Diversions	18,537	4,566	23,103
Water Treatment Plant	-	10,000	10,000
Permanent Camp	27,878	-	27,878
Total Onsite Infrastructure	149,245	39,937	189,182

The auxiliary facilities include a variety of offices, shops, and warehouses that support the day-to-day operations of the mine and the plant. Table 21.8 lists the main auxiliary facilities and their direct costs that were included in the initial CAPEX.

Table 21.8: Onsite Auxiliary Facilities CAPEX

Onsite Auxiliary Facilities	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Ancillaries - General	56	-	56
Administration Building	2,240	-	2,240
Security Building	1,588	-	1,588
Medical & Emergency Services	2,332	-	2,332
Mine Ops/Mine Dry Building	1,889	-	1,889
Warehouse	5,466	-	5,466
Truck Shop/Truck Wash/Truck Warehouse	15,227	-	15,227
Reagents Storage	2,129	-	2,129
Plant Maintenance Building	4,350	-	4,350
Assay Lab	470	-	470
Fuel Station	2,637	-	2,637
Explosives Storage	167	-	167
Total Onsite Auxiliary Facilities	38,552	-	38,552

The capital components that make-up the tailings management system consist of the tailings dam, the tailings impoundment, tailings pumps, slurry pipeline system, water reclaim system, TSF under-liner drains, TSF surface water diversions, and the civil work that is required to route the tailings and reclaim water lines between the process plant and the TSF. The water reclaim system consists of reclaim barge, pumps, pipeline, and storage tank. The TSF impoundment will be expanded three times during the life-of-mine. Due to limited availability of construction material during preproduction, the initial TSF dam will be constructed to a lower level (Stage 1A) than the rest of the impoundment (Stage 1B), and then raised to the Stage 1B level during the first two years of production. Including the Stage 1B raise, the tailings dam is planned to be raised four times during the life-of-mine. Table 21.9 summarizes the direct costs for the TSF.

Table 21.9: Tailings Storage Facility CAPEX

Tailings Storage Facility	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Tailings Dam	5,309	4,861	10,170
Tailings Impoundment	17,340	18,179	35,520
Tailings & Water Reclaim	20,107	2,331	21,591
TSF and Reclaim Total	42,757	25,371	67,280

The water diversions include the surface diversions that divert non-contact water around the TSF and WRSF, a sedimentation pond located below the surface water diversions, the surface approaches and exit to the tunnel water diversion beneath the Yellow Pine pit, and the tunnel water diversion itself. The surface water diversions were estimated by Tierra Group International while the tunnel diversion was estimated by Cementation USA Inc., an underground contractor. Initial CAPEX includes the initial diversions around the TSF and tunnel diversion around the Yellow Pine open pit; the sustaining CAPEX is for additional diversions associated with the TSF, as shown in Table 21.10.

Table 21.10: Surface Water Diversion CAPEX

Water Diversions	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Starter TSF/WRSF Diversions	2,566	-	2,566
EFSFSR Diversion Tunnel Approaches	1,987	-	1,987
EFMC Sedimentation Pond (lined)	149	-	149
Initial WRSF SWM Pond (unlined)	106	-	106
East Fork Diversion Tunnel (Underground)	13,729	-	13,729
EFMC Rock Drain & Outlet	-	223	223
Final WRSF SWM Pond (lined)	-	262	262
Stage 3 TSF/WRSF Diversions	-	3,784	3,784
Hangar Flats diversion	-	296	296
Water Diversion Total	18,537	4,566	23,103

The 300-bed operations camp would be formed from the 1,000-bed construction camp by removing 700 beds after start-up; the dining and housekeeping facilities, fresh water supply, power distribution, and wastewater treatment at the camp would remain. The direct costs are based mainly on budgetary quotations of from a local supplier of modular camps with specific experience at the Stibnite Gold site. The cost per bed equates roughly to $40,000 per bed. The cost to furnish the camp was estimated to be 10% of the cost of the buildings. The total direct cost of the operations plus construction camp facility, shown in Table 21.7, does not include the cost of catering or housekeeping.

21.1.4.2 Offsite Infrastructure

The offsite infrastructure includes three main components: the power transmission line, the mine access road, and the Cascade Complex that includes administration offices, the production assay lab, the staging area for mine personnel transportation, and some warehousing capacity. Table 21.11 summarizes the direct costs estimated for these three components.

Table 21.11: Offsite Infrastructure Summary

Off-Site Infrastructure	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Power-line	40,224	-	40,224
Mine Access Roads	34,302	-	34,302
Off-Site Cascade Complex	5,801	-	5,801
Total Off-site Infrastructure	80,327	-	80,327

Midas Gold's Cascade Complex is described in Section 18.5. The estimated direct costs of these facilities, shown in Table 21.11, do not include land acquisition costs for the facility; these costs are included in owner's costs.

The power transmission line and substation upgrades are described in Section 18. The cost for the power transmission line was developed by Power Engineers Inc. of Boise, Idaho, in consultation with Idaho Power Company. The power transmission line includes upgrading of seven substations between Emmett, Idaho and McCall, Idaho to handle the new power demands. Table 21.12 summarizes the Power Engineers cost estimate including indirect costs.

Table 21.12: Power Transmission Construction and Substation Upgrades

Component	Power Transmission Line Costs	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Direct Costs	Power Infrastructure Improvements and Construction	40,224	-	40,224
Indirect Costs	Mob/Demob	420	-	420
	Construction Management	2,293	-	2,293
	Engineering Cost	4,172	-	4,172
Power Transmission Line Totals		47,109	-	47,109

The mine access road is described in Section 18. The cost was developed by HDR Inc.; the cost estimate includes civil excavation costs, placement of aggregate base course and geotextile, emplacement of culverts, retaining walls, installation/upgrade of bridges, the installation of a storm water drainage system, and other minor costs. HDR also estimated the indirect costs and contingency. Table 21.13 summarizes HDR's cost estimate.

Table 21.13: Mine Access Road CAPEX

Component	Mine Access Road Costs	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Direct Costs	Mine Road Design and Construction	34,302	-	34,302
Indirect Costs	Mob/Demob	2,046	-	2,046
	Other Indirect Costs	6,543	-	6,543
Mine Access Road Totals		42,890	-	42,890

21.1.5 Indirect Costs

Indirect costs are those costs that can generally not be tied to a specific work area, as summarized in Table 21.14. This category includes "other direct costs" that are related to construction that can't be assigned directly to a work area including the following:

- quality assurance testing is included at 2% of total direct costs for civil, concrete, piping, steel, and electrical costs;
- survey is included at 1% of total direct costs for civil, concrete, and steel costs;

- mobilization of contractors is 0.5% of total direct cost without mine & mobile equipment and including quality assurance;
- pipe spooling detail is included at 3% of piping materials; and
- programming included at 0.2% of direct costs.

Table 21.14: Indirect Capital Cost Summary

Indirect Cost Items	Cost ($000s)
Quality Assurance Testing	6,184
Surveying	1,526
Pipe Spooling	696
Programming	967
Construction Camp Costs	18,726
Freight	22,580
EPCM Costs	81,915
Start-up and Commissioning	1,990
Vendor Erection Supervision	1,194
Capital and Commissioning Spares	4,974
Other Indirect Costs	23,700
Sales Tax	12,233
Total Indirect Costs	176,687

21.1.5.1 EPCM Costs

M3 breaks down estimated EPCM costs into various categories that total 16.7% of direct constructed field cost excluding mining pre-strip and mine equipment costs, as shown in Table 21.15.

Table 21.15: EPCM Capital Cost Summary

EPCM Components	Percentage of Total Direct Field Cost	Cost ($000s)
Temporary Facilities & Support	0.5%	2,499
Project Services	1.0%	4,997
Project Control	0.75%	3,748
Management & Accounting	0.75%	3,748
EPCM Fee Fixed	1.0%	5,183
EPCM Construction Trailers	0.2%	999
Engineering	6.0%	27,758
Construction Power	0.1%	500
Construction Management	6.5%	32,483
EPCM Total	16.7%	81,915

21.1.5.2 Other Indirect Costs

Table 21.14 also includes indirect costs from other consultants for infrastructure construction, including the power transmission line, mine access road, TSF, and water diversions. The indirect costs for these tasks were provided by the estimating entity, as detailed in Table 21.16.

Table 21.16: Other Indirect Capital Costs

Other Indirect Costs		Cost ($000s)
Process Plant Indirect Costs	Mob/Demob	2,659
Power Transmission Line Indirect Costs	Power-line Mob/Demob	420
	Power-line Construction Management	2,293
	Power-line Engineering	4,172
Mine Access Road Indirect Costs	Access Road Mob/Demob	2,046
	Other Access Road Indirect Costs	6,543
TSF Indirect Costs	TSF Mob/Demob	300
	TSF/Portal Indirect Costs	1,400
Diversion Tunnel Indirect Costs	Diversion Tunnel Indirect Costs	3,867
Total Other Indirect Costs		23,700

21.1.6 Owner Costs

Owner costs were developed to cover specific functions relating to the construction of the Project. Owner costs exclude exploration and corporate costs and are summarized in Table 21.17.

Key staff, plant and equipment operators will be hired as early as three months prior to start-up for training, and preparation work. Senior staff and engineering personnel will also be hired several months prior to start-up as they become available. Environmental monitoring will continue through the construction period. Other Owner Cost items include:

- Owner's construction and administrative costs, including the Owners camp;
- plant mobile equipment cost;
- insurance, accounting and legal;
- furniture and office equipment;
- tools;
- staffing and operator training cost; and
- initial fills and wear steel spares.

Table 21.17: Owner Team Capital Costs

Owner Team Item	Sub Section	Total ($000s)
Owner Team Salaries & Burden	Construction Management Team	9,330
Owner Team Indirect Costs	Phone	100
	Radio	100
	IT Hardware	200
	Software	500
	Medical and Safety Supplies	130
	Housing	375
	Food	1,095
	Offices & Furniture	90

Owner Team Item	Sub Section	Total ($000s)
Environmental, Social & Permitting Consulting Support	Permitting	2,000
	Compliance	800
	Legal	750
	Community Relations	180
Land Acquisition	Land for Offsite Complex	500
Insurance	Construction Insurance	2,000
Operations Staff Build-Up & Training	Administration Team	200
	Mine Team	800
	Process Team	600
	Job-Specific Training	1,500
Operations Direct Costs	Small Tools	225
	Light Vehicles	2,931
	Warehouse First Fills	2,400
Total Owner Costs		26,806

21.1.7 Environmental Remediation & Mitigation Costs

21.1.7.1 Environmental Remediation

Environmental remediation costs include readily identifiable components of the Project that are related to relocating specific historical materials to new location and include the spent heap leach ore stored in the SODA area and on the Hecla heap pad. The costs for these remediation projects have been included in the mining costs; as noted in Table 21.1, Table 21.2 and Table 21.5. A breakdown of the environmental remediation costs is summarized in Table 21.18.

Table 21.18: Environmental Remediation Cost Summary

Environmental Remediation Items	Initial CAPEX ($000s)	OPEX ($000s)	Total Remediation Costs ($000s)
SODA Relocation	1,694	3,952	5,646
Hecla Heap Relocation	566	1,320	1,885
Totals	2,259	5,272	7,531
Note: All costs included in this table are included in mining CAPEX and OPEX.			

Not included in the environmental remediation costs are the following, with reasons identified:

- Removal and reprocessing of the 3.0 Mst of Historic Tailings in the Indicated Mineral Resource category since these materials are above cut-off grade and therefore treated as Mineral Reserves in the economic model, with costs treated as OPEX against the revenue received.
- Removal and relocation of historical waste rock material within the footprint of the Yellow Pine and West End open pits, since these materials are inadequately defined to determine precise quantities and are therefore aggregated into the overburden mining costs and included in the OPEX. As noted in Section 25, there is an opportunity based on limited available data that a component of these materials may have sufficient grade to reprocess as ore, which would increase revenues and reduce strip ratios and therefore reduce OPEX.

- Remediation of the EFMC (Blowout Creek) drainage, which is primarily addressed by placing several million tons of newly mined and suitable waste rock to buttress the slope thereby reducing erosion combined with sediment control during operations and a newly developed surface channel at closure. The placement of this material is included in the waste material costs and included in OPEX.
- Remediation of historically impacted areas (e.g. historical mill and smelter site, tailings areas, town sites, shop areas, haul roads, heap leach facilities, etc.) where new facilities are to be located in the current Project design (e.g. Hangar Flats open pit, Main WRSF, process plant area, new haul roads, truck shop, etc.) are not separated out as a mitigation cost, rather they are incorporated into the construction cost of the new facilities and the subsequent closure costs for the these areas affected by the Project included in Closure Costs.
- Restoration of historically impacted drainages, wetlands, slopes, as well as replanting of areas historically affected by forest fires is all captured in Closure Costs.

21.1.7.2 Stream and Wetland Mitigation

Stream and wetland mitigation costs are estimated from expected impacts to those resources as identified in jurisdictional delineations conducted during baseline evaluations. A substantial effort was made during the Project design phase to minimize impacts to both jurisdictional wetlands and stream reaches through facilities design, siting, and orientation. Additionally, as the site has been impacted significantly by historic operations, facilities siting criteria were designed and implemented to minimize impacts to previously un-disturbed wetlands and stream channels.

Recognizing that impact mitigation will be a negotiated process with multiple agencies and stakeholder input, and that a Functional Assessment of wetland conditions and function has not yet been completed, unit costs were developed from western US mitigation projects and known planned disturbances to establish an estimate of the anticipated mitigation costs for the Project. Costs for mitigation have been allocated in time to capture anticipated development schedules and planned disturbance recognizing that development timing occurs after mitigation as appropriate.

Table 21.19: Stream and Wetland Mitigation Costs

Item	Initial ($000s)	Sustaining ($000s)
Wetland Mitigation Costs	4,265	3,370
Stream Mitigation Costs	6,341	4,795
Totals	10,606	8,165

21.1.8 Closure Costs

Closure costs were developed utilizing the Standardized Reclamation Cost Estimator (SRCE) a tool that was developed for mining projects in the State of Nevada and currently supported directly by the Nevada Division of Environmental Protection. This model is used extensively by public and private property mining projects in multiple states and jurisdictions. The inputs for the model are updated annually for unit costs and, where applicable, user-edited data files were included to account for Idaho and Project specific items.

Reclamation specific costs are developed from the SRCE model utilizing specific design elements detailed in the Project's operation and closure plans. Costs were then incorporated into the overall project cost model in the year that they occur.

Closure costs include items such as potential long-term water treatment, reclamation maintenance activities, and long-term site monitoring which may include surface and ground water monitoring, vegetation success monitoring, aquatic species and habitat monitoring, and chemical and physical stability.

The basis for cost components of the long-term closure and monitoring activities are factored from anticipated operational costs, experience from closure operations of similar projects, and standard unit costs. The schedule of costs for reclamation, closure, and post-closure are allocated along the life of mine and closure based upon expected reclamation and closure related activities.

Table 21.20: Closure Costs

Item	Initial ($000s)	Sustaining ($000s)	Closure ($000s)	Total Closure ($000s)
Cost for Arranging Bond	762	7,052	3,230	11,044
Earthwork / Re-contouring	-	2,133	15,808	17,941
Re-vegetation / Stabilization	-	-	2,395	2,395
Detoxification / Water Treatment/Disposal of Wastes	-	-	15,008	15,008
Structure, Equipment and Facility Removal, and Misc.	-	-	4,510	4,510
Monitoring	-	-	1,202	1,202
Construction Management & Support	-	-	6,473	6,473
Indirect Costs	-	-	7,915	7,915
Totals	762	9,185	56,541	66,488

21.1.9 Contingency

Contingency costs, as summarized in Table 21.21, are estimates of the costs that are not included in the CAPEX that can be expected to be spent during construction. The more engineering that is done ahead of the estimate the higher the accuracy of the CAPEX and thus, the lower the contingency costs. The total estimated contingency for this Project is 17.2% of the total initial CAPEX before contingency and is considered typical for a prefeasibility-level study.

Table 21.21: Summary of Contingency Capital Costs

Contingency Capital Costs	Cost ($000s)
Mine Capital - 5%	2,378
Plant, Auxiliaries, Cascade Facilities - 17.5%	105,990
Power Transmission Line - 15%	7,070
Access Road - 30%	12,867
TSF - 15%	3,397
Water Diversions - Surface 15%	721
Water Diversions - Underground 17.5%	3,079
Owner's Cost - 17.5%	4,691
Mitigation Costs - 17.5%	1,856
Total Contingency	142,050

21.2 OPERATING COST

The average cash operating cost before by-product credits, royalties, refining and transportation charges over the LOM and during the first four years of operations are estimated to be $26.65/st and $27.15/st of ore processed, respectively. The average cash operating cost after by-product credits but before royalties, refining and transportation charges over the LOM and during the first four years of operations are estimated to be $23.20/st and $21.83/st of ore processed, respectively. These cash costs include mine operations, process plant operations, and general and administrative costs (G&A) and are summarized in Table 21.22.

Table 21.22: OPEX Summary

Cash Operating Cost Estimate	LOM Average			Years 1-4 Average	
	$/st mined	$/st milled	$/oz Au	$/st milled	$/oz Au
Mining OPEX(1)	2.00	9.08	222	10.04	222
Processing OPEX	-	14.45	354	14.10	312
General & Administrative OPEX	-	3.13	77	3.01	67
Cash Costs Before By-Product Credits(2)	-	26.65	653	27.15	601
By-Product Credits	-	-3.45	-85	-5.32	-118
Cash Costs After of By-Product Credits(2)	-	23.20	568	21.83	483

Note:
(1) Mining OPEX excludes capitalized stripping.
(2) Cash costs shown in this table are before royalties, refining, and transportation charges. Cash costs that include these costs are presented in Section 22.

Major cost items driving the OPEX estimate include power (diesel and electricity), reagents (lime, sodium metabisulfite, cyanide, and copper sulfate), and labor. The details that comprise the OPEX are provided in the sections that follow.

21.2.1 Major Reagents, Fuel and Electricity Costs

Table 21.23 summarizes the unit costs for the major Project consumables (process reagents, diesel fuel and power). A more detailed list of the consumables for the Project is provided in Table 21.30.

Table 21.23: Cost Assumptions for Reagents and Power

Item	Unit	Cost Estimate	Comment
Diesel fuel	$ per gallon	3.28	Quote for off-road diesel to site
Electricity	$ per kWhr	0.05876	Price rate quote
Lime	$ per ton	253	Price quote to site
Sodium Cyanide	$ per lb	1.134	Price quote to site
Sodium Metabisulfite	$ per lb	0.295	Price quote to site
Copper Sulfate	$ per lb	1.32	Price quote to site

21.2.2 Labor Requirements

Labor for the Project was estimated for the mine, process plant, and G&A support. Labor rates were estimated using market surveys for the region and comparable wage rates from other mining operations in the area. Onsite personnel were assumed to be housed in a camp facility and working 12-hour shifts on a 14-day on, 14-day off work schedule except for salaried employees. A breakdown of the labor requirements stratified by function (mine, process, or G&A) and location (onsite or offsite) is presented in Table 21.24 with the annual estimated payroll for an average year.

Table 21.24: Estimated Labor Requirements

Labor Category	Number of Personnel			Average Annual Payroll ($000s)
	Low	Peak	Average	
Mine Operations Personnel - Hourly	62	153	121	9,130
Mine Personnel - Salaried	26	27	26	2,988
Mine Maintenance Personnel - Hourly	45	77	62	5,311
Mine Maintenance Personnel - Salaried	8	9	9	1,012
Process Operations Personnel - Hourly	72	72	72	6,199
Process Operations Personnel - Salaried	13	13	13	1,467
Process Maintenance Personnel - Hourly	56	56	56	4,968
Process Maintenance Personnel - Salaried	5	5	5	558
G&A Hourly Personnel - Onsite	62	62	62	4,506
G&A Salaried Personnel - Onsite	9	9	9	1,112
G&A Hourly Personnel - Offsite	28	28	28	2,020
G&A Salaried Personnel - Offsite	12	12	12	1,491
Labor Totals	398	523	475	40,762

21.2.3 Mine Operating Cost

Mine operating costs were developed based on first principles for the mine plan and equipment list presented in Section 16. The unit costs for labor were jointly developed by Midas Gold and M3. Fuel costs were set at $3.28 per gallon. Table 21.25 summarizes the consumable and labor operating costs by the unit operations.

Table 21.25: Life-of-Mine Mining Cost Averages

Mining Function	Percentage	Unit Cost ($/st)
Drilling	9.4%	0.168
Blasting	10.0%	0.179
Loading	9.7%	0.172
Hauling	38.4%	0.685
Auxiliary	16.6%	0.296
General Mine	5.0%	0.090
General Maintenance	4.6%	0.082
G&A	6.3%	0.113
Total for Material Mined	100.0%	1.786
Drilled and Blasted		2.004
Mill Feed		7.943

The operating costs have been broken into quarterly time periods for preproduction and years 1 and 2 to parallel the mine plan. Preproduction is established to be 15 months or 5 quarters. During the first quarter of preproduction (Qtr -5), the costs shown are for development of the initial access roads to the mine working areas. No in-pit mining tonnage is moved during that period so there is no calculation of “cost per ton”. The cost per ton in all remaining periods is based on the total in situ tonnage mined from the pits within the mine plan. Preproduction development costs (Table 21.3) are carried 30% as CAPEX and the remaining 70% as OPEX. Table 21.26 summarizes the total mine operating cost per time period. This table should provide a clear indication of the mine operating costs by year of operation. The major mining fleet is planned to be leased. Equipment down payments and monthly payments will be treated as operating costs in the economic analysis. Therefore, down payments and monthly payments are shown as operating costs in Table 21.26.

Table 21.26: Mine OPEX by Period

Period	Consumables and Labor Operating Cost ($000s)	Equipment Lease and Down Payments ($000s)	Totals ($000s)
PP Q-5	-	-	1,999
PP Q-4	-	-	3,332
PP Q-3	-	-	5,937
PP Q-2	-	-	6,188
PP Q-1	-	-	7,648
Yr 1 Q1	10,125	3,191	13,316
Yr 1 Q2	12,178	4,127	16,305
Yr 1 Q3	12,899	3,197	16,096
Yr 1 Q4	13,152	3,197	16,349
Yr 2 Q1	14,056	4,037	18,093
Yr 2 Q2	15,095	5,859	20,954
Yr 2 Q3	17,574	7,427	25,001
Yr 2 Q4	17,970	5,237	23,207
Yr 3	73,532	18,559	92,091
Yr 4	79,372	20,427	99,799
Yr 5	76,746	11,428	88,174
Yr 6	75,713	7,274	82,987
Yr 7	62,296	6,787	69,083
Yr 8	64,292	4,286	68,578
Yr 9	66,129	4,252	70,381
Yr 10	60,229	5,565	65,794
Yr 11	43,041	4,844	47,885
Yr 12	28,695	2,108	30,803
Totals	743,094	121,802	890,000

Note: Operating Costs shown in pre-production are the 70% of preproduction consumable and operating costs allocated to OPEX.

The mine operating costs provided in Table 21.25 include:

1. Drilling, blasting, loading, and hauling of material from the mine to the crusher, stockpiles or waste storage facilities. Maintenance of the waste storage areas and stockpiles is included in the mining costs. Maintenance of mine mobile equipment is included in the operating costs.
2. Mine supervision, mine engineering, geology and ore control are included in the G&A category.
3. Operating labor and maintenance labor for the mine mobile equipment are included.
4. Mine access road construction and maintenance is included. If mine haul trucks drive on the road, its cost and maintenance is included in the mine operating costs.

5. Relocation of SODA material and reprocessing of Historic Tailings is included.
6. Delivery of mine waste rock to the tailings dam construction is included. However, placement and compaction of that material at the TSF is not included.
7. The small stockpile (572 kst) that is generated during preproduction stripping would be fed to the crusher in Year 1.
8. The cost of backfilling the Yellow Pine open pit with West End waste rock is included.
9. A general mine allowance is included that is intended to cover mine pumping costs and general operating supplies that cannot be assigned to one of the unit operations.
10. A general maintenance allowance is included that is intended to cover the general operating supplies of the maintenance group.

The mine is planned to work two 12 hour shifts per day for 365 days per year. Ten days (20 shifts) of lost time are assumed due to weather delays.

21.2.4 Plant Operating Cost

This section addresses the following operating costs process plant OPEX. The process plant operating costs are summarized by the categories of labor, electric power, liners (wear steel), grinding media, reagents, maintenance parts and services, annual POX shutdown, oxygen, and supplies and services, as presented in Table 21.27.

Table 21.27: Process Plant OPEX Summary by Category

Plant Operation Cost Category	LOM Cost ($000s)	Cost ($/st)
Labor	158,316	1.61
Power	205,550	2.10
Liners	45,465	0.46
Grinding Media	122,336	1.25
Reagents	581,955	5.93
Maintenance Parts & Services	123,598	1.26
Annual POX Shutdown	52,000	0.53
Oxygen	85,361	0.87
Supplies & Services	42,229	0.43
Totals	1,416,809	14.45

The processing costs divided by process area are provided in Table 21.28

Table 21.28: Process Plant OPEX by Process Area

Process Area	LOM Cost ($000s)	Cost ($/st)
Crushing and Conveying	24,857	0.25
Grinding & Classification	395,942	4.04
Antimony Recovery	39,813	0.41
Gold Flotation	130,895	1.33
Pressure Oxidation	202,382	2.06
CCD and Neutralization	146,633	1.50
CIP Leaching and Cyanide Recovery & Detox	347,873	3.55
Carbon Handling & Refinery	45,685	0.47
Tailings & Water Reclaim	25,736	0.26
Water Treatment	4,151	0.04
Fresh Water System	4,091	0.04
Ancillaries	48,752	0.50
Total Process Plant	1,416,809	14.45

21.2.4.1 Process Plant Labor Cost

The process plant operating and maintenance labor costs were derived from a staffing plan and are based on labor rates from an industry survey for this region and modified where necessary. The annual salaries include overtime and benefits for both salaried and hourly employees. The burden rate used is 35% for hourly staff and 40% for salaried staff to include a 5% average annual bonus. The process labor numbers of personnel and costs divided by process area are provided in Table 21.29.

Table 21.29: Process Labor Costs by Process Area

Labor Costs by Process Area	Number of Personnel	Annual Labor Cost ($000s)
Crushing and Conveying	8	691
Grinding	12	1,080
Antimony Recovery	8	691
Gold Flotation Operator	4	378
Pressure Oxidation	12	1,004
Tailings	4	346
Carbon Handling & Refinery	12	1,037
Ancillaries	25	2,439
Maintenance	61	5,526
Totals	146	13,193

21.2.4.2 Reagents

Reagent consumption rates were determined from the metallurgical test data or industry practice. Budget quotations were received for reagents supplied from local sources where available, with an allowance for freight to site or from historical data from other projects. Estimated LOM reagent costs by process area are presented in Table 21.30.

Table 21.30: LOM Reagent Costs by Process Area

Process Area	Reagent	Life-of-Mine ($000s)
Grinding	Lime	735
	Sodium Cyanide	2,069
	Copper Sulfate	44,252
Antimony Recovery	Lead Nitrate	10,850
	Aerophine 3418A	2,196
	Methyl Isobutyl Carbinol	1,996
	Sodium Cyanide	552
Gold Flotation	Copper Sulfate	23,919
	Potassium Amyl Xanthate	39,042
	Methyl Isobutyl Carbinol	21,618
Pressure Oxidation	Flocculant	1,075
CCD and Neutralization	Flocculant	9,390
	Lime	95,835
CIP Leaching and Cyanide Recovery & Detoxification	Sodium Cyanide	98,066
	Lime	99,585
	Carbon	17,836
	Sodium Metabisulfite	94,279
	Copper Sulfate	1,557
Carbon Handling & Refinery	Sodium Hydroxide	1,933
	Nitric Acid	15,169
Total LOM Reagent Cost		581,955

21.2.4.3 Maintenance Wear Parts and Consumables

Wear parts consumption (liners) and grinding media were estimated on a pound/ton basis. The consumption rate and unit costs were used to calculate the annual costs and cost per unit of production. These consumption rates and costs are shown in Table 21.31.

Table 21.31: Life-of-Mine Wear Steel Cost

Wear Steel Category	Applicable Equipment	Life-of-Mine Costs ($000s)
Liners	Primary Crusher	1,049
	Pebble Crusher	1,061
	SAG Mill	31,149
	Ball Mill	12,205
Grinding Media	SAG Mill	31,285
	Ball Mill	91,051
Total LOM Wear Steel Cost		167,801

An allowance was made to cover the cost of maintenance for the facilities and all items not specifically identified. The allowance made as a percent of the direct capital cost of equipment for each area; the rate used was 5%.

21.2.5 General and Administrative Cost

General and Administration costs include management, accounting, human resources, environmental and safety compliance, laboratory, community relations, site residential camp, communications, insurance, legal, training, and other costs not associated with either mining or processing. The LOM G&A cost estimated for the Project are presented in Table 21.32.

Table 21.32: General and Administration Cost Detail

Cost Item	LOM ($000s)
Labor & Fringes (G&A and laboratory)	130,781
Accounting (excluding labor)	1,200
Safety (excluding labor)	1,200
Human Resources (excluding labor)	1,200
Environmental Department (excluding labor)	6,000
Security (excluding labor)	1,800
Laboratory (excluding labor)	9,000
Janitorial Services (contract)	2,400
Community Relations (excluding labor)	9,000
Office Operating Supplies and Postage	3,000
Maintenance Supplies	9,565
Maintenance Labor, Fringes, and Allocations	3,000
Power	2,190
Propane	2,400
Phone/Communications	3,600
Licenses, Fees, and Vehicle Taxes	1,800
Legal	15,000
Insurances	42,000
Subs, Dues, Public Relations, and Donations	1,200
Travel, Lodging, and Meals	3,600
Camp (excluding labor)	54,000
Training	3,000
Total	306,936

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22 ECONOMIC ANALYSIS

The economic analysis presented in this Report uses a financial model that estimates cash flows on an annual basis for the life of the Project at the level of detail appropriate to the prefeasibility level of engineering and design. Annual cash flow projections are estimated over the LOM based on the CAPEX, OPEX, sales revenue and other cost estimates. CAPEX is estimated in four categories: initial, sustaining, closure and reclamation, and working, and are distributed in accordance with the estimated year of expenditure. OPEX estimates include labor, reagents, maintenance, supplies, services, and electrical power for each year. The sales revenue is based on payable metals contained in doré bullion and antimony concentrate produced by the process. Other costs, such as royalties, taxes, and depreciation are estimated in accordance with the present stage of the Project.

The financial model results are presented in terms of Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the Project. Annual cash flow projections are estimated over the LOM based on the estimates of capital expenditures and production cost and sales revenue. The estimates of CAPEX and OPEX have been developed specifically for this Project, as presented in Section 21.

22.1 ASSUMPTIONS

Assumptions that were used to estimate the CAPEX and OPEX are presented in Section 21. Specific assumptions used in the construction of the financial model are provided below.

- A discount rate of 5% is applied to NPV calculations.
- Funding for the Project is assumed to be 100% equity funding with no financing costs except leasing of major mining equipment since this equipment would almost certainly be lease purchased.
- Revenue for doré and antimony concentrates is claimed in the same year as it is produced.
- Costs incurred prior to the construction commencement date are not included in the model and are considered “sunk costs,” except for tax purposes, where the aggregate expenditures accumulated prior to the construction commencement date are available to offset taxes.
- A 15-day delay in revenue from sales and a 15-day delay in payment of accounts payable are used in the formulation of working capital, which is recaptured at the end of mine life.
- An allowance of 5% is included in the financial model for salvage value of selected capital equipment, excluding buildings and tanks, which are included in the reclamation costs.
- Depreciation is calculated using the Modified Accelerated Cost Recovery System (MACRS) method in accordance with current U.S. Internal Revenue Service (IRS) regulations.
- Depletion is estimated for the financial model using the percentage method; a rate of 15% is used for gold and silver and 22% is used for antimony.

22.2 REVENUE

Revenue for the financial model is based on the grade and tonnage of mill feed from the mine plan (Table 22.1), using the plant recovery for the specific mineralization type to yield metal production figures (Table 22.2). The appropriate refinery or smelter treatment terms (Table 22.3) are applied to the payable metals (Table 22.4) using the metal prices presented in Table 22.5. Metal prices were fixed in mid-2014 for mine planning purposes.

Table 22.1: Life of Mine Contained Metal by Deposit

Deposit	Ore Type	Ore Tons (kst)	Contained Metal Grade			Contained Metal Quantity		
			Gold (oz/st)	Silver (oz/st)	Antimony (%)	Gold (oz)	Silver (oz)	Antimony (klb)
Yellow Pine	High Sb	6,750	0.065	0.210	0.593	438,012	1,420,491	80,012
	Low Sb	37,235	0.056	0.069	0.009	2,085,266	2,552,639	6,364
Hangar Flats	High Sb	4,284	0.056	0.166	0.425	238,068	711,929	36,438
	Low Sb	11,146	0.040	0.055	0.019	449,227	614,683	4,320
West End	Oxide	10,736	0.022	0.029	-	233,295	215,682	-
	Low Sb	24,914	0.041	0.044	-	1,023,994	1,089,717	-
Historical Tailings	High Sb	3,001	0.034	0.084	0.165	101,315	251,920	9,906
Totals / Averages		98,066	0.047	0.071	0.070	4,569,176	6,857,061	137,040

Table 22.2: Recovered Metal Production

Deposit	Doré Bullion		Antimony Concentrate		
	Gold (koz)	Silver (koz)	Antimony (klb)	Gold (koz)	Silver (koz)
Yellow Pine	2,263	338	69,822	12	611
Hangar Flats	597	68	30,030	5	349
West End	1,090	681	-	-	-
Historic Tailings[1]	72	20	-	-	-
Totals Production	4,023	1,107	99,852	17	960

Annual metal production by deposit is illustrated on Figure 22.1.

Figure 22.1: Annual Metal Production by Deposit

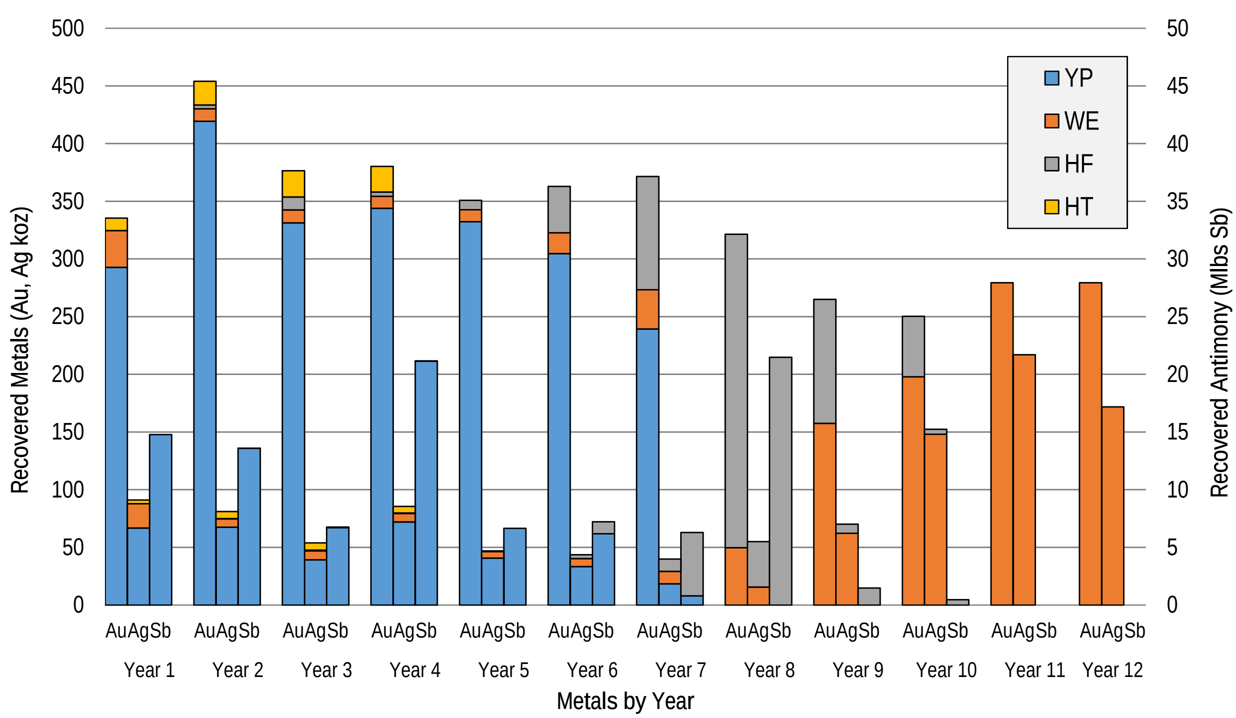


Table 22.3: Smelter Treatment Factors

Gold and Silver Bullion	
Gold Payability	99.5%
Silver Payability	98.0%
Refining Charge – Au (per troy ounce)	$1.00
Transportation Charge – Au (per troy ounce)	$1.15
Refining Charge – Ag (per troy ounce)	$0.50
Transportation Charge – Ag (per troy ounce)	$1.15
Antimony Concentrate	
Payable Antimony (%)	68%
Gold Payability (approximate)	
<5.0 g/t	0%
5.0 to <8.5 g/t	15-20%
8.5 to <10.0 g/t	20-25%
≥10.0 g/t	25%
Silver Payability (approximate)	
<300 g/t	0%
300 to <700 g/t	40-50%
≥700 g/t	50%
Transportation to Asia (per wet ton)	$151

Table 22.4: Payable Metals Production

Product	Gold (koz)	Silver (koz)	Antimony (klb)
Doré Bullion	4,002	1,085	-
Antimony Concentrate	3	382	67,900
Total Payable Metals	4,006	1,467	67,900

Table 22.5: Metal Price Cases

Case	Metal Prices			Basis
	Gold ($/oz)	Silver(1) ($/oz)	Antimony(1) ($/lb)	
Case A	1,200	20.00	4.00	Lower-bound case that reflects the lower prices over the past 36 months and spot on December 1, 2014.
Case B (Base Case)	1,350	22.50	4.50	Approximate 24-month trailing average gold price as of December 1, 2014.
Case C	1,500	25.00	5.00	Approximate 48-month trailing average gold price as of December 1, 2014.
Case D	1,650	27.50	5.50	An upside case to show Project potential at metal prices approximately 20% higher than the base case.

Note:
(1) Prices were set at a constant gold:silver ratio ($/oz:$/oz) of 60:1 and a constant gold:antimony ratio ($/oz:$/lb) of 300:1 for simplicity of analysis, although individual price relationships may not be as directly correlated over time. Historic gold:silver ratios have averaged around 60:1.

22.3 CAPITAL COSTS

The details of the CAPEX estimate for the Project are summarized below and are presented in more detail in Section 21. For purposes of the financial model, CAPEX is broken into four categories: initial capital, sustaining capital, closure and reclamation capital, and working capital. Table 22.6 presents a summary of the initial, sustaining and closure and reclamation capital costs.

Table 22.6: Capital Cost Summary

Capital Cost Area	Initial CAPEX ($000s)	Sustaining CAPEX ($000s)	Closure CAPEX ($000s)	Total CAPEX ($000s)
Direct Costs	613,343	76,862	-	690,206
Indirect Costs	176,687	4,275	-	180,962
Owner's Costs	26,806	-	-	26,806
Environmental Mitigation Costs	10,606	8,165	-	18,771
Closure Bonding, Closure and Reclamation Costs	762	9,185	56,542	66,489
Contingency	142,050	-	-	142,050
Totals	970,254	98,488	56,542	1,125,283

22.3.1 Initial Capital

The total initial CAPEX carried in the financial model for new construction and pre-production mine development is expended over a 3 year period. The initial CAPEX includes direct and indirect capital costs, owner's costs and contingency. The initial CAPEX would be expended in the years before production and a small amount carried over into the first production year.

22.3.2 Sustaining Capital

A schedule of CAPEX incurred during the production period was estimated and included in the financial analysis under the category of sustaining capital. The LOM sustaining capital is estimated to be $98.5 million, as shown in Table 22.6. This capital will be expended over a 12-year period.

22.3.3 Reclamation and Closure

Reclamation and closure costs were estimated to be $56.5 million on a gross basis. The estimate does not include approximately $102.4 million in gross payable revenue from the 75 koz of gold to be recovered from Historic Tailings as part of the Project legacy clean-up, nor does it include savings incurred from using the 7.3 million tons of spent heap leach ore in TSF construction, which is material that would otherwise have had to be obtained from other sources at additional cost.

22.3.4 Working Capital

A 15-day delay of receipt of revenue from sales is used for accounts receivable. A delay of payment for accounts payable of 15 days is also incorporated into the financial model. Working capital is estimated to be $7.5 million before production and an additional $18 million immediately after commencement of production but prior to receipt of revenue. Working capital also includes an allowance for capital tied up in parts inventory prior to its use. All the working capital is recaptured at the end of the mine life and the final value of these accounts is $0.

22.4 OPERATING COSTS

The average cash operating cost before by-product credits, royalties, refining and transportation charges over the LOM and during the first four years of operations are estimated to be $26.65/st and $27.15/st of ore processed, respectively. The average cash operating cost after by-product credits but before royalties, refining and transportation charges over the LOM and during the first four years of operations are estimated to be $23.20/st and $21.83/st of ore processed, respectively. These cash costs include mine operations, process plant operations, and general and administrative costs (G&A) and are summarized in Table 22.7.

Table 22.7: Operating Cost Summary

Cash Operating Cost Estimate	LOM Average			Years 1-4 Average	
	$/st mined	$/st milled	$/oz Au	$/st milled	$/oz Au
Mining OPEX(1)	2.00	9.08	222	10.04	222
Processing OPEX	-	14.45	354	14.10	312
General & Administrative OPEX	-	3.13	77	3.01	67
Cash Costs Before By-Product Credits(2)	-	26.65	653	27.15	601
By-Product Credits	-	-3.45	-85	-5.32	-118
Cash Costs After of By-Product Credits(2)	-	23.20	568	21.83	483

Note:
(1) Mining OPEX excludes capitalized stripping.
(2) Cash costs shown in this table are before royalties, refining, and transportation charges; cash costs that include these costs are presented in Table 22.8.

22.5 ROYALTIES, DEPRECIATION AND DEPLETION

There is a 1.7% royalty that applies to gold revenue, as detailed in Section 4. The LOM reduction in Net Operating Income is estimated to be $91.9 million.

Depreciation is calculated using the MACRS method starting with the first year of production. The initial capital and sustaining capital used a 7 year life. The last year of production is the catch up year for the assets that are not fully depreciated at that time.

The percentage depletion method was used in the evaluation. It is determined as a percentage of gross income from the property, not to exceed 50% of taxable income before the depletion deduction. A rate of 15% is used for gold and silver and a rate of 22% is used for antimony.

22.6 TAXATION

22.6.1 Income Tax

Taxable income for income tax purposes is defined as metal revenues minus operating expenses, royalty, property and severance taxes, reclamation and closure expense, depreciation and depletion. Deduction for depletion is used in the calculation of State income tax, but no deduction is taken for the federal income taxes paid. The combined effective tax rate was calculated as follows:

$$\begin{aligned}\text{Combined Effective Tax Rate} &= \text{State Rate} + \text{Federal Rate} \times (100\% - \text{State Rate})\\ &= 7.4\% + 35\% \times (100\% - 7.4\%)\\ &= 39.8\%\end{aligned}$$

22.6.2 Idaho Mine License Tax

This is a tax for the privilege of mining or receiving royalties from mining operations. The tax rate is 1% of the value of ores mined or extracted and royalties received. The basis is the taxable income that is defined by the IRS.

22.7 TOTAL PRODUCTION COSTS

A detailed breakdown of the various measures of cash cost over the life of the mine are shown in Table 22.8. The costs are presented in $/st mined, $/st milled, and in $/oz Au. The table provides the cash costs before and after by-product credits for the LOM and initial four years of operation, as well as the total cash costs, which include royalties, refining and transportation charges for the LOM and initial seven years of operation and, finally, the All in Costs (AIC) that includes non-sustaining capital.

Table 22.8: Total Production Cost Summary

Total Production Cost Item	LOM			Years 1-4	
	($/st mined)	($/st milled)	($/oz Au)	($/st milled)	($/oz Au)
Mining	2.00	9.08	222	10.04	222
Processing	-	14.45	354	14.10	312
G&A	-	3.13	77	3.01	67
Cash Costs Before By-Product Credits	-	26.65	653	27.15	601
By-Product Credits	-	-3.45	-85	-5.32	-118
Cash Costs After of By-Product Credits	-	23.20	568	21.83	483
Royalties	-	0.94	23	0.34	23
Refining and Transportation	-	0.25	6	1.04	8
Total Cash Costs	-	24.38	597	23.20	513
Sustaining CAPEX	-	1.00	24	0.52	11
Salvage	-	-0.27	-7	0.00	0
Property Taxes	-	0.04	1	0.04	1
All-In Sustaining Costs	-	25.15	616	23.76	526
Reclamation and Closure(1)	-	0.58	14	-	-
Initial (non-sustaining) CAPEX(2)	-	9.89	242	-	-
All-In Costs	-	35.62	872	-	-

Notes:
(1) Defined as non-sustaining reclamation and closure costs in the post-operations period.
(2) Initial Capital includes capitalized preproduction.

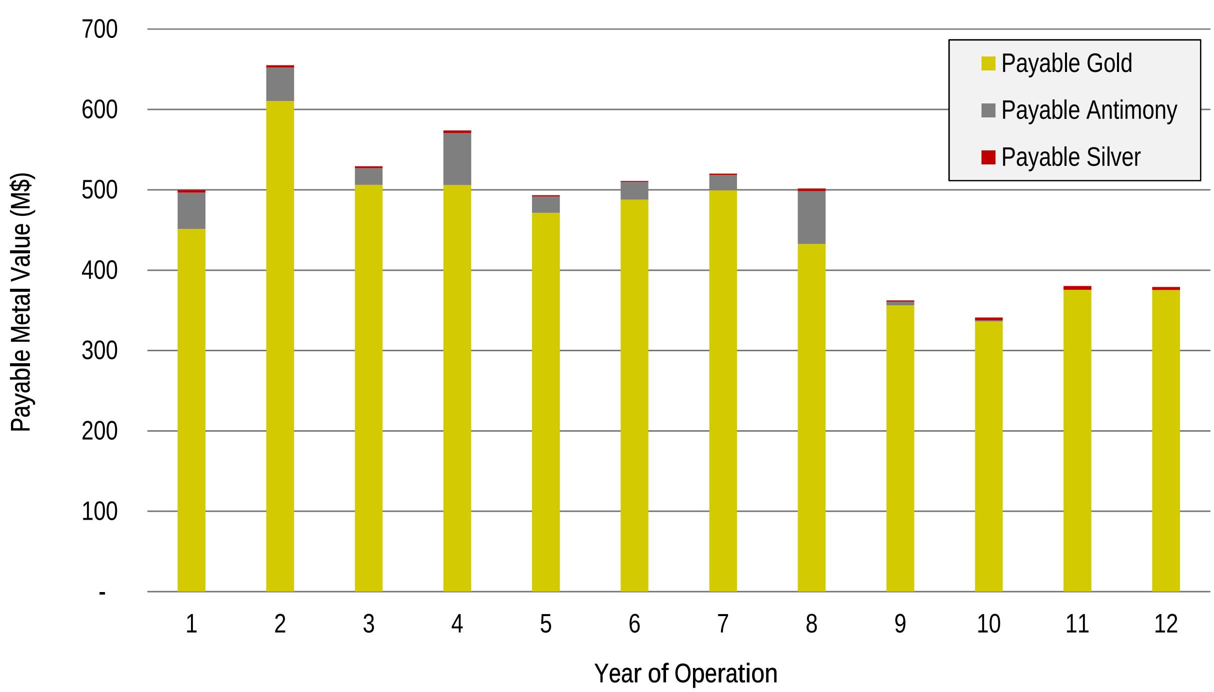
22.8 FINANCIAL MODEL RESULTS

The financial model results are presented in terms of NPV, IRR, and payback period in years for recovery of the capital expenditures. These economic indicators are presented on both pre-tax and after-tax bases. The NPV is presented both undiscounted ($NPV_{0\%}$) and at a 5% discount rate ($NPV_{5\%}$), as shown in Table 22.9, and indicates that on an after-tax basis the Project has an $NPV_{5\%}$ of $832 million, an IRR of 19.3%, and a payback period of 3.4 years.

Table 22.9: Financial Model Pre-Tax and After-Tax Indicators by Case

Parameter	Unit	Pre-tax Results	After-tax Results
Case A ($1,200/oz Au, $20.00/oz Ag, $4.00/lb Sb)			
$NPV_{0\%}$	M$	1,286	1,041
$NPV_{5\%}$	M$	662	513
IRR	%	16.2	14.4
Payback Period	Production Years	4.0	4.1
Case B ($1,350/oz Au, $22.50/oz Ag, $4.50/lb Sb)			
$NPV_{0\%}$	M$	1,915	1,499
$NPV_{5\%}$	M$	1,093	832
IRR	%	22.0	19.3
Payback Period	Production Years	3.2	3.4
Case C ($1,500/oz Au, $25.00/oz Ag, $5.00/lb Sb)			
$NPV_{0\%}$	M$	2,543	1,929
$NPV_{5\%}$	M$	1,524	1,129
IRR	%	27.2	23.4
Payback Period	Production Years	2.6	2.9
Case D ($1,650/oz Au, $27.50/oz Ag, $5.50/lb Sb)			
$NPV_{0\%}$	M$	3,171	2,344
$NPV_{5\%}$	M$	1,955	1,414
IRR	%	31.9	27.0
Payback Period	Production Years	2.2	2.5

Figure 22.2: Payable Metal Value by Year for Case B in Millions of Dollars

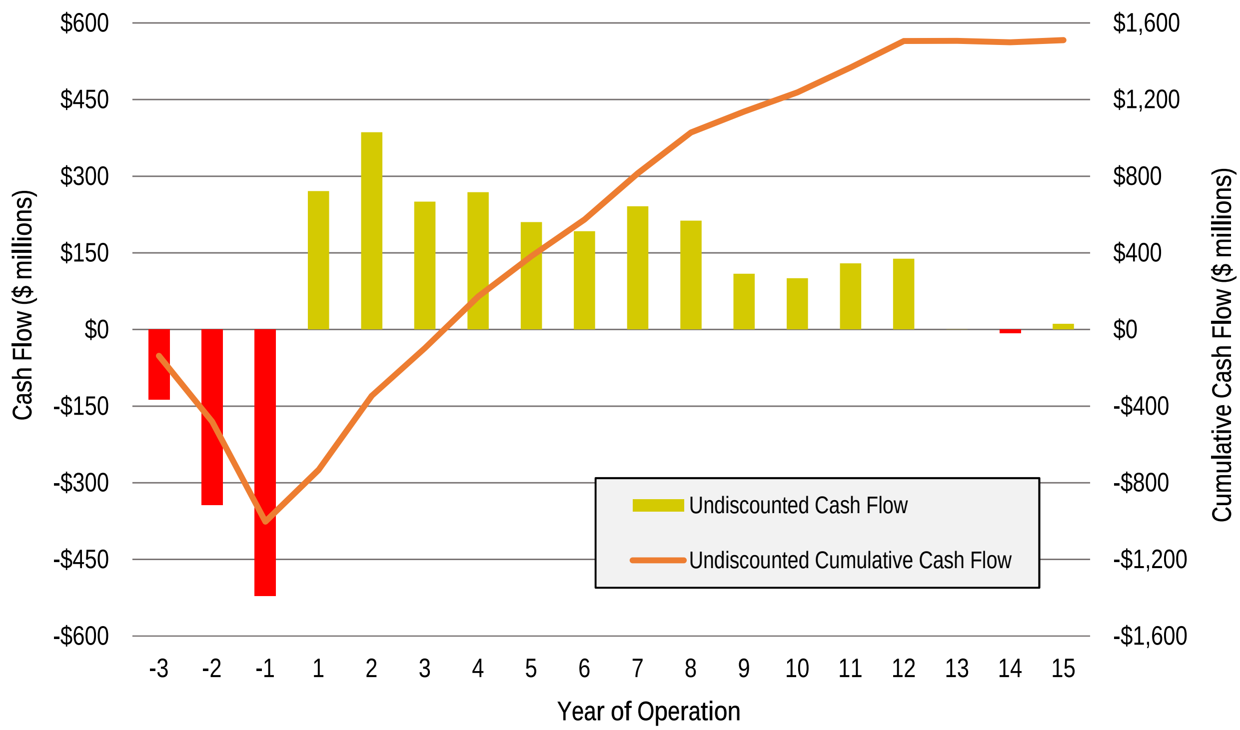


The after-tax, undiscounted payback periods for each case are as follows.

- Case A: 4.1 production years
- Case B: 3.4 production years
- Case C: 2.9 production years
- Case D: 2.5 production years

The undiscounted cash flows for Case B, the base case, are depicted on Figure 22.3.

Figure 22.3: Undiscounted After-Tax Cash Flow for Case B



22.9 MINE LIFE

Using the current Mineral Reserve and the nominal design throughput of 22,050 stpd, the mine plan projects a 12-year production life. Construction is projected to require a three-year period after the permits are obtained and prior to the start of commercial operations. Closure is projected to take at least 10 years post-production, with some reclamation work occurring concurrently with operations, and the bulk of the closure activities and costs incurred in the first 3 years after operations cease. Some closure activities and long-term monitoring are anticipated to continue well after the reclamation period is complete to ensure that the closure designs continue to protect the environment and are performing in accordance with the design parameters.

22.10 SENSITIVITY ANALYSIS

The sensitivity of the financial model was tested with respect to metal prices or gold grade, initial CAPEX, and OPEX for each case. The value of each parameter was raised and lowered 20% to evaluate the impact of such changes on the NPV at a 5% discount rate. The results for the pre-tax $NPV_{5\%}$ are presented in Table 22.10, and the results for after-tax $NPV_{5\%}$ are presented in Table 22.11. After-tax sensitivities with respect to $NPV_{0\%}$, $NPV_{5\%}$, IRR, and payback in production years are presented in Table 22.12.

Table 22.10: Pre-tax $NPV_{5\%}$ Sensitivities by Case

Case	Variable	Pre-tax $NPV_{5\%}$ (M$)		
		-20% Variance	0% Variance	20% Variance
Case A	CAPEX	862	662	463
	OPEX	1,017	662	308
	Metal Price or Grade	-27	662	1,352
Case B (Base Case)	CAPEX	1,292	1,093	894
	OPEX	1,447	1,093	739
	Metal Price or Grade	318	1,093	1,869
Case C	CAPEX	1,723	1,524	1,325
	OPEX	1,878	1,524	1,170
	Metal Price or Grade	662	1,524	2,386
Case D	CAPEX	2,154	1,955	1,755
	OPEX	2,309	1,955	1,600
	Metal Price or Grade	1,007	1,955	2,902

Table 22.11: After-tax $NPV_{5\%}$ Sensitivities by Case

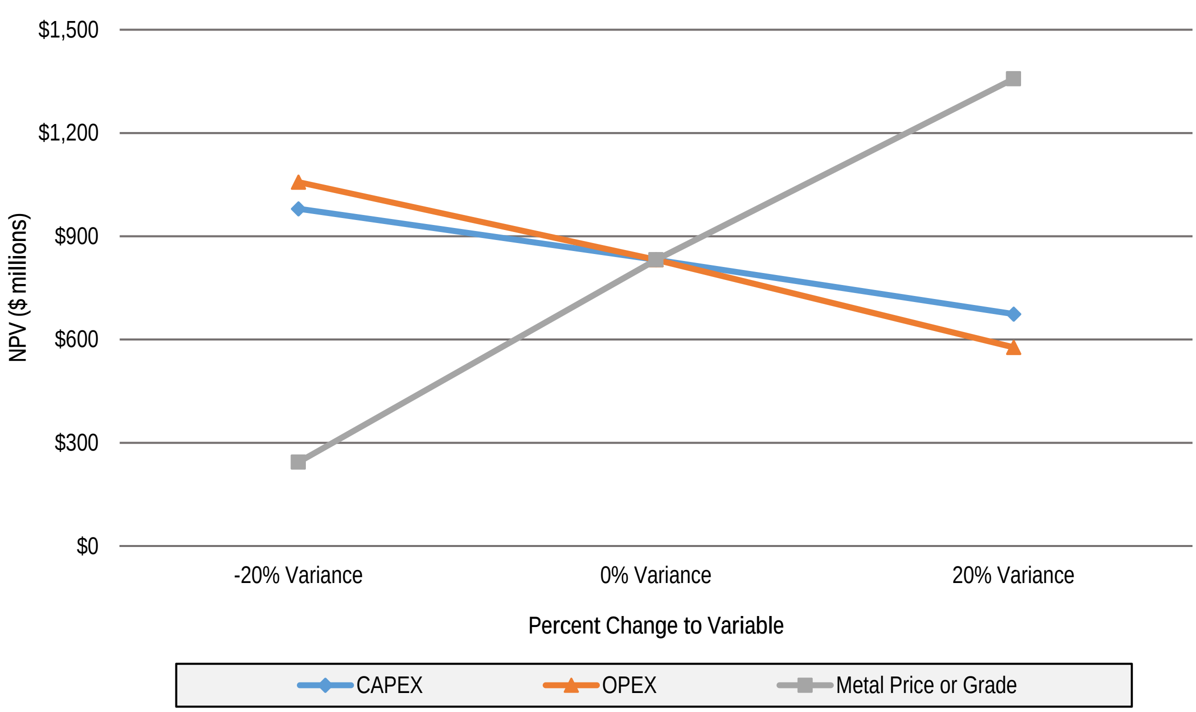
Case	Variable	After-Tax $NPV_{5\%}$ (M$)		
		-20% Variance	0% Variance	20% Variance
Case A	CAPEX	676	513	346
	OPEX	760	513	239
	Metal Price or Grade	-30	513	1,012
Case B (Base Case)	CAPEX	980	832	674
	OPEX	1,057	832	577
	Metal Price or Grade	244	832	1,357
Case C	CAPEX	1,266	1,129	982
	OPEX	1,341	1,129	903
	Metal Price or Grade	513	1,129	1,696
Case D	CAPEX	1,548	1,414	1,277
	OPEX	1,623	1,414	1,200
	Metal Price or Grade	770	1,414	2,035

Table 22.12: Base Case After-Tax Sensitivity Analysis

Variance	$NPV_{0\%}$ (M$)	$NPV_{5\%}$ (M$)	IRR (%)	Payback (yrs)
Metal Prices or Gold Grade				
20%	2,262	1,357	26.3	2.6
10%	1,887	1,100	23.0	2.9
0%	1,499	832	19.3	3.4
-10%	1,089	546	15.0	4.0
-20%	656	244	9.8	5.4
Capital Cost				
20%	1,332	674	15.2	4.0
10%	1,417	754	17.1	3.7
0%	1,499	832	19.3	3.4
-10%	1,578	907	21.8	3.0
-20%	1,654	980	24.7	2.7
Operating Cost				
20%	1,130	577	15.5	3.9
10%	1,323	710	17.5	3.6
0%	1,499	832	19.3	3.4
-10%	1,665	946	20.9	3.2
-20%	1,828	1,057	22.4	3.0

The after-tax sensitivities for $NPV_{5\%}$ (Table 22.12) for Case B are illustrated on Figure 22.4.

Figure 22.4: Case B After-Tax $NPV_{5\%}$ Sensitivities



The $ATNPV_{5\%}$ of the Project is most sensitive to changes in revenue, which is manifested as changes in metal prices and gold grades. For example, a 20% increase in gold price or gold grade leads raises the $ATNPV_{5\%}$ from $832 million to $1,357 million, a 63% increase. Similarly, a decrease of 20% in gold grade or gold price results in a 71% decrease in $ATNPV_{5\%}$.

All of the cases indicate that the Project is a bit more sensitive to changes in OPEX than it is to changes in CAPEX. For example, the change in $ATNPV_{5\%}$ for a 20% increase in CAPEX is -19%, where as a 20% increase in OPEX causes a -31% change in $ATNPV_{5\%}$.

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23 ADJACENT PROPERTIES

The Project is not impacted by adjacent properties. No data or information from adjacent properties was used to support this PFS.

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24 OTHER RELEVANT DATA AND INFORMATION

24.1 ANTIMONY INFORMATION

24.1.1 Introduction

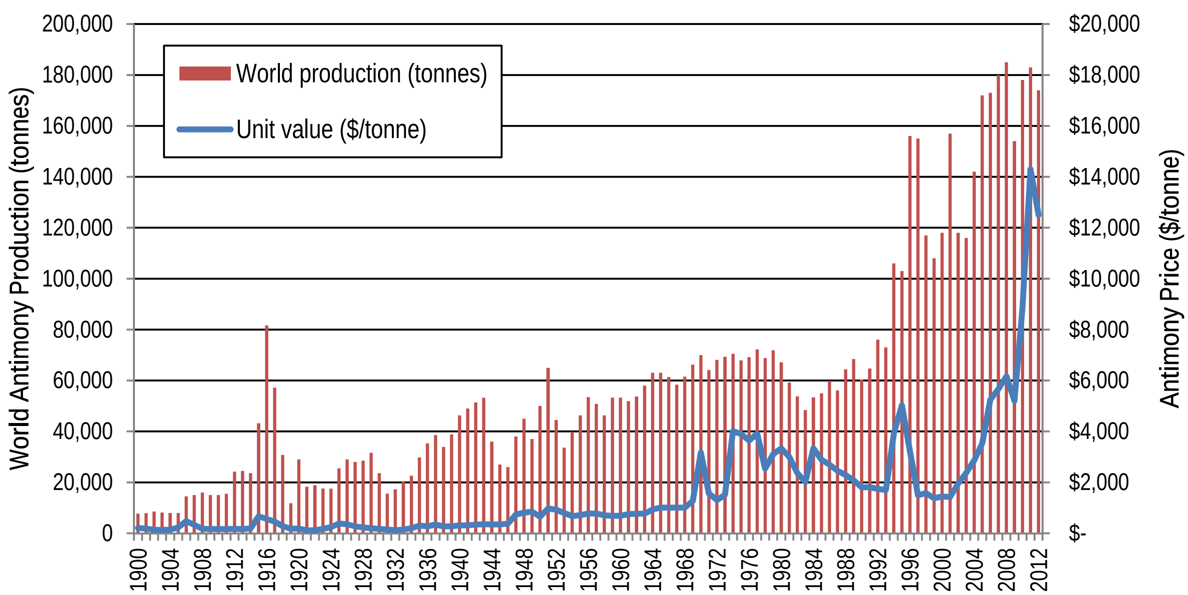
The name "antimony" is derived from the Greek meaning "never found alone". The principal use of antimony today is as an oxide synergist in the flame retardant chemical additive sector. Antimony (Sb) is a silvery-white, shining, soft and brittle metal. It is a semiconductor and has thermal conductivity lower than most metals. Due to its poor mechanical properties, pure antimony is only used in very small quantities; larger amounts are used for alloys and in antimony compounds.

24.1.2 History

China has dominated world supply for the past 110 years; the most famous deposit in China is the Hsikwangshan deposit in Hunan, reputedly worked since the 16th Century to become a world-class producer and it is still the dominant source, producing 30,000 to 40,000 tonnes of contained antimony per annum.

From 1897 to 1911, the average world production of antimony metal was just over 10,000 tonnes with an average metal price of 7.5¢/lb ($165/tonne). From 1911 to 1914, production increased from 15,000 tonnes per annum to 22,000 tonnes per annum, with prices remaining at similar levels to those before. During World War I, production rose sharply to 82,000 tonnes in 1916 as the metal's physical properties for ammunition were deemed important. Metal prices rose to a peak of 32¢/lb in 1915 and settled back down to 8¢/lb after the end of the war, but remained volatile between 5¢/lb (ammunition stockpile destocking) and 19¢/lb for the rest of the decade. Peacetime demand declined to around 22,000 tonnes per year with the US consuming ~10,000 tonnes per year, with a further 5,600 tonnes per year from recycling ore as a by-product from lead ore. Metal prices jumped once more during the Korean war of 1950 - 53, reaching over $1,000/st for the first time in the history of the metal. China dramatically increased its production in the late 1980s and 1990s to command 90% of production once more (summarized from Tri-Star, 2012). Figure 24.1 illustrates antimony production and pricing since 1900.

Figure 24.1: World Antimony Production and Price from 1900 – 2012



Source: USGS, 2014b

24.1.3 Supply

Of the ~200,000 tonnes of contained antimony presently produced annually on a worldwide basis, approximately one-quarter is from secondary production via recycling of antimony bearing metal alloys. Of the balance, three-quarters of that is produced as primary antimony, while approximately 10% is produced from antimony bearing residues from lead smelting. As such, approximately 135,000 tonnes per annum of antimony is produced from antimony concentrates and ores (Confidential Report, 2014). As can be seen from the table below, China remains by far the world's largest producer of primary antimony.

Table 24.1: Estimated Mine Production of Antimony by Country

Country	Annual Antimony Production (tonnes)					
	2008	2009	2010	2011	2012e	2012 (%)
Australia	1,500	1,000	1,106	1,577	2,481	1.4%
Bolivia	3,905	2,990	4,980	3,947	4,000	2.3%
Canada	132	64	9000	10000	7,000	4.0%
China	166,000	140,000	150,000	150,000	145,000	83.3%
Kyrgyzstan	700	700	700	1,500	1,500	0.9%
Peru	531	145	-	-	-	0.0%
Russia	3,500	3,500	6,040	6,348	6,500	3.7%
South Africa	3,983	2,673	3,700	4,700	3,800	2.2%
Tajikistan	2,000	2,000	2,000	2,000	2,000	1.1%
Turkey	2,700	1,400	650	3,400	1,900	1.1%
Totals	185,000	154,000	178,000	183,000	174,000	100.0%
Source: USGS, 2013						

Six companies, including Hunan Hsikwangshan, Guangxi China Tin and Hunan Chenzhou Mining account for 90% of China's supply (Roskill 2012), accounting for around 70% of official mine production in 2011, down from around 80% in earlier years. According to Chinese government statistics, over 75% of all of China's reported primary antimony production in 2013 was from Hunan province, followed far behind by Guangxi and Yunnan, however, government statistics underreport and are adjusted regularly without explanation, making analysis challenging. However, Hsikwangshan Twinkling Star Company Limited (Twinkling Star) is acknowledged as the world's largest integrated antimony producer. Located in Lengshuijiang, it produces ~30% of antimony products in China and ~25% globally. Capacity is ~32,000 tonnes per annum of contained antimony metal plus trioxide. Twinkling Star is a state-owned company; its parent corporation is the Hunan Nonferrous Group, which itself is majority owned by China Minmetals Corporation (Minmetals). Chenzhou Mining Company Limited (Chenzhou) is an integrated antimony and gold producer, producing approximately 19,000 tonnes of antimony products per annum and 6 tonnes of gold. As their own mine contains significant amounts of gold, Chenzhou operates a recovery circuit to capture gold from antimony smelting and separate it using an electrowinning refining process. Chenzhou Mining is majority owned by the Hunan Gold Group (aka Hunan Jinxin Gold Group) and is therefore considered a state-owned company. Other large operations in China include Multi Antimony Corp. (4,000 tonnes), China Tin Group (~4,000 tonnes), Guangxi Youngsun Metals (~10,000 tonnes) plus, following a government imposed consolidation, there are nine smelters (apart from Twinkling Star's operation) in Lengshujiang, each with a minimum capacity of 5,000 tonnes per annum (Confidential Report, 2014).

China appears unwilling (if not unable) to maintain its level of mine production given resource depletion, rising costs, environmental crackdown, and resource conservation (Confidential Report, 2014). As a result, production in China is unlikely to increase over the next few years and could even fall in the face of government determination to limit environmental damage from smaller operations (Roskill, 2012). However, the rate of fall may be slower than was forecast by Roskill in 2012 (Confidential Report, 2014).

Other sources of supply outside China include Geopromining in Russia, producing 5,000-6,000 tonnes of contained antimony in gold-antimony concentrates, and Mandalay Resources producing approximately 6,000 tonnes of contained antimony in gold-antimony concentrates. Comsup Commodities Inc. (via Anzob LCC) owns the Jizhikrut antimony-mercury deposit in Tajikistan; the mine is estimated to be producing at a rate of 4,500 tonnes per annum of contained antimony albeit with high mercury (0.6%-1.0%) with a new smelter that was built in 2013. The Consolidated Murchison mines in South Africa have been operating since the 1930s, with production of ~2,200 contained tonnes antimony in 2013; with the closure of their roaster, ConsMurch now sells its concentrates to China and India. Suspended operations include Beaver Brook in Canada, which is owned by Twinkling Star and is estimated to have two years of reserves left, and Hillgrove in New South Wales, Australia, which has a potential production capacity of 4,000 to 5,000 tonnes of contained antimony per year.

Roskill (2012) comments that new capacity could enter the market to meet growing demand but notes that increased production elsewhere is likely to offset any declines in Chinese production in the short term. Roskill has identified a number of significant additional sources of antimony concentrates in Europe, N. America, Africa and Oceania that could add over 14,000 tonnes per year Sb to world mine capacity within the next four years. However, non-China antimony deposits thus far identified are insufficient to keep up with demand increases (Confidential Report, 2014).

Based on the forecast for demand growth and China's falling production, it is estimated that an additional 18,000 tonnes of annual primary mine production will need to be brought online through 2030 to meet demand (Confidential Report, 2014).

24.1.4 Critical Minerals Status

In 2013, the U.S. Department of Defense (DoD) ranked antimony #2 in the list of strategic and non-fuel defense material shortfalls (US DoD, 2013) and foresees a shortfall of 20,500 tons in a four-year period, and recommends mitigation options to address this shortfall including strategic stockpiling of ~11,000 tons of antimony.

Also in 2013, the US Geological Survey (USGS, 2013) estimated world primary mine production at 167,000 tonnes, of which 150,000 tonnes came from China (a 95% dependence ratio), with Bolivia in a distant second at 4,980 tonnes. World refined production was estimated by the USGS in the same report at 194,510 tonnes, of which 187,000 tonnes came from China (a 98% dependence ratio), with Bolivia again a distant second at 2,200 tonnes.

In a report on critical raw materials for the European Union (2014), the European Commission identifies an estimated 93% of the supply and estimates that all but 6% (i.e. 87% of the world's supply) comes from China. Out of the 19 critical raw materials identified in the European Commission report (2014), antimony is the only metal ranked as being in deficit in the three time horizons evaluated (2012, 2015 and 2020), estimating a small deficit in 2012 and forecasting a large deficit in 2015 and 2020.

In 2012, the British Geological Survey ranked antimony with a relative supply risk index of 9.0, the second highest risk ranking of the 41 commodities considered due to China's dominance of world production and reserves, compounded by the relatively low level of recycling and low substitutability.

24.1.5 Stockpiling

China's State Reserve Bureau (SRB) has been active in recent years buying antimony metal in China, purchasing approximately 10,000 tonnes in 2013. The total amount of antimony stockpiled by the SRB is unknown. Apart from the material held by the SRB, Minmetals (as of 2014) was understood to have maintained a stockpile of

20,000 tonnes of metal and oxide in Guangxi warehouses. Moving forward, it is unknown if the SRB or Minmetals will continue to purchase surplus production (Confidential Report, 2014).

A private exchange, the Fanya Minor Metals Exchange, that functions as a quasi-ETF began warehousing minor metals in 2014, holding 2,600 tonnes of antimony as of 30 June 2014 (Confidential Report, 2014).

24.1.6 Smelting and Refining

Outside of China, integration of mining and smelting or downstream processing has become rarer as previously integrated operations have found it difficult to compete with Chinese production. For that matter, there is little smelter production of any scale outside of China. Tri-Star Resources and US Antimony Corp. (Montana) are attempting to integrate mining supply with processing facilities, but are in relatively early stages of development. Tri-Star intends to begin construction of its smelter in Oman in 2015. There are a number of facilities in Belgium, France, Bolivia and India producing primary trioxide and recycling antimony from lead acid batteries; outside of Bolivia, none produce antimony from mines.

China has been increasingly importing antimony concentrates since 2007, with imports of concentrates (not contained metal) increasing from 17,000 tonnes in 2007 to more than 68,000 tonnes in 2012 (Minmetals, 2013) and ~64,500 tonnes in 2013 (Confidential Report, 2014). Looking at contained metals, Chinese imports of antimony in concentrates are estimated at ~25,500 tonnes in 2013, led by Russia (~8,300 tonnes of contained antimony in 2013), Australia (~5,300 tonnes), Tajikistan (~4,400 tonnes) and Myanmar (~2,800 tonnes), although smuggled imports are likely much higher from Myanmar (Confidential Report, 2014).

24.1.7 Export Quotas

China has imposed export quotas for antimony and antimony products since 2009; the table below summarizes the announced quotas and amount actually exported during the year.

Table 24.2: Chinese Antimony Export Quotas

Export Quota Component	2011	2012	2013	2014
Total Annual Quota Announced as Headline Figure	60,300	59,400	59,400	59,400
Actual Quota Released to Each	59,502	58,584	58,562	58,512
Actual Exports During Year	40,128	43,482	28,524	N/A
Unused Quota	20,172	15,918	30,876	N/A
Percent of Quota Utilized	67%	74%	49%	N/A

24.1.8 Primary Antimony Uses

The largest use of antimony oxide is in a synergistic system with a halogen (generally chlorine or bromine) flame retardant system for plastics and textiles. Normal applications for this product include upholstered chairs, rugs, television cabinets, business machine housings, electrical cable insulation, laminates, coatings, adhesives, circuit boards, electrical appliances, seat covers, car interiors, tape, aircraft interiors, fiberglass products, carpeting, etc. Around 90% of flame retardant production ends up in electronics and plastics, while the remaining 10% ends up in coated fabrics and furniture upholstery and bedding.

The principal uses of antimony outside of flame retardant include:

- an alloy in lead-acid batteries;
- military equipment and ammunitions;

- alternative wind and solar energy applications involving fire resistant transmission lines;
- a catalyst in food applications such as plastic packaging and water bottles.

Antimony is also used as a decolorizing agent in optical glass such as photocopiers, camera lenses, binoculars and IPad screens. It is also used in semi-conductors and many components of motor vehicles. Antimony oxide is used as a phosphorescent agent in fluorescent light bulbs and antimony oxide, antimony trisulphide and/or organic compounds of antimony are added to fluid lubricants and/or molybdenum disulphide to improve performance.

One potential key future growth area could be in computer phase change memory, which is projected to lead to 1 gigahertz transfer speeds (30x faster than flash) (Visual Capitalist, 2012).

24.1.9 Reserves

Global reserves are estimated at 1.8 million tonnes (USGS, 2014a) which, at an estimated global production rate of 0.2 million tonnes per year (Roskill, 2012), is estimated to be less than 9 years of production. Furthermore, while official Chinese statistics still report considerable reserves, independent estimates suggest that they might be reaching exhaustion, particularly in the area of Lengshuijiang City, the center of antimony mining in China. Although some resources were discovered in 2011, very few deposits have been explored or developed in recent years (Roskill, 2012).

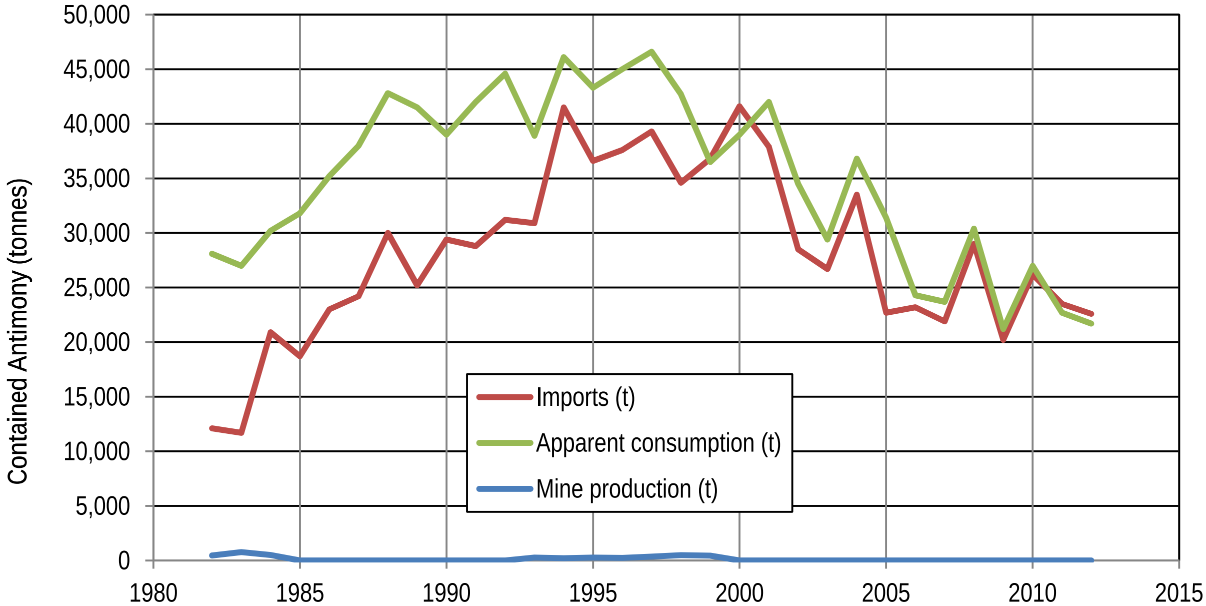
Table 24.3: World Mine Production and Reserves of Antimony

Country	Mine Production of Antimony (tonnes)		Antimony Reserves (tonnes)
United States	-	-	-
Bolivia	4,000	5,000	310,000
China	145,000	130,000	950,000
Russia (recoverable)	6,500	6,500	350,000
South Africa	3,800	4,200	27,000
Tajikistan	2,000	4,700	50,000
Other countries	13,000	13,000	150,000
World Totals (rounded)	**174,000**	**163,000**	**1,800,000**
Source: USGS, 2014a			

24.1.10 US Perspective

Historically, the US used to meet a significant amount of its demand from domestic mine production, but imports began to climb rapidly in the early 1980s from ~12,000 tonnes in 1982 to ~24,000 tonnes five years later, and passing 35,000 tonnes 1995 before reaching a peak of 41,600 tonnes in 2000. Imports have fallen to ~22,000 tonnes in 2012 as the result of the effects of rising prices, the global financial crisis and substitution. According to the USGS (2014a), there was zero domestic mine production in 2013, and there is one processing facility in Montana producing minor amounts of antimony metal and oxide from imported feedstock; as a result, US dependence on imports is 100%. For 2013, the USGS estimates US consumption of 24,000 tonnes of contained antimony, and US imports were estimated at 25,000 tonnes. Sources of import were: China, 71%; Mexico, 9%; Belgium, 8%; Bolivia, 5%; and other, 7%. The Mexican imports are the source of the feedstock for the plant in Montana; Belgian imports come from a processing facility there that imports feedstock from elsewhere.

Figure 24.2: US Production, Imports & Consumption of Antimony



Source: USGS, 2014b

The estimated US domestic distribution of primary antimony consumption was as follows: metal products, including antimonial lead (for batteries) and ammunition, 35%; nonmetal products, including ceramics and glass and rubber products, 35%; and flame retardants, 30% (USGS, 2014b).

24.1.11 Outlook

According to Roskill (2012), growth in consumption has been led by high growth rates in Asia, particularly in China, over the past few years.

Overall, antimony demand remains highly dependent on the level of consumption of antimony trioxide in the flame-retardants sector and antimony metal in lead-acid batteries; Roskill (2012) estimates that these two sectors accounted for nearly 80% of antimony consumption worldwide in 2011.

Non-metallurgical markets for antimony are forecast by Roskill (2012) to increase by nearly 4% per year through to 2016 with higher growth for flame-retardants, plastic catalysts and heat stabilizers tempered by lower growth in ceramics and other uses. Other sources suggest a growth rate in the range of 1.5% per annum (Confidential Report, 2014).

According to Roskill (2012), metallurgical markets are forecast to increase by nearly 2% per year, as the antimony content of new lead-acid batteries continues to fall. Lead alloys will show higher rates of growth because of increasing usage in construction applications in emerging economies.

Continued growth in demand for antimony, especially trioxide, combined with the uncertainty over the ability of China to increase production because of resource and environmental limitations, means prices are likely to stay high and volatile (Roskill, 2012). Prices for antimony trioxide could rise to $15,000 per tonne by 2016 (in 2011 dollar terms), eclipsing the $13,000 per tonne peak witnessed in 2011 (Roskill, 2012). Subsequent to the Roskill report of 2012, which was essentially issued at the peak of the antimony prices, antimony substitutes may have drawn some demand away from antimony demand and this may explain some of the recent softening in the antimony prices.

Others are more conservative on their outlook, given reduced demand resulting from slowing economies and the increase in antimony substitutes, with the recent high prices and production not falling as fast as expected in China (Confidential Report, 2014) and forecast that antimony prices will be range bound at $8,000-$11,000 per tonne until current surplus is consumed, likely through 2018, and do not see prices being sustained above $13,000 per tonne even during the forecast deficit period post-2020, depending on the rate of demand growth, how quickly Chinese production falls and the development of additional production. Minmetals (2013) forecast volatile prices as markets adjust to changing conditions, but commented that prices should go up.

24.2 SECONDARY ANTIMONY PROCESSING

The process design and flowsheet developed for this PFS were establish based on producing a by-product antimony concentrate with sale of the concentrate to an antimony smelter (all suitable currently operating antimony smelters are located in Asia). This approach was considered appropriate given the estimated cost and perceived complexity of building and operating a secondary antimony processing plant; however, several compelling advantages were identified that support further study of secondary processing of the antimony concentrate including:

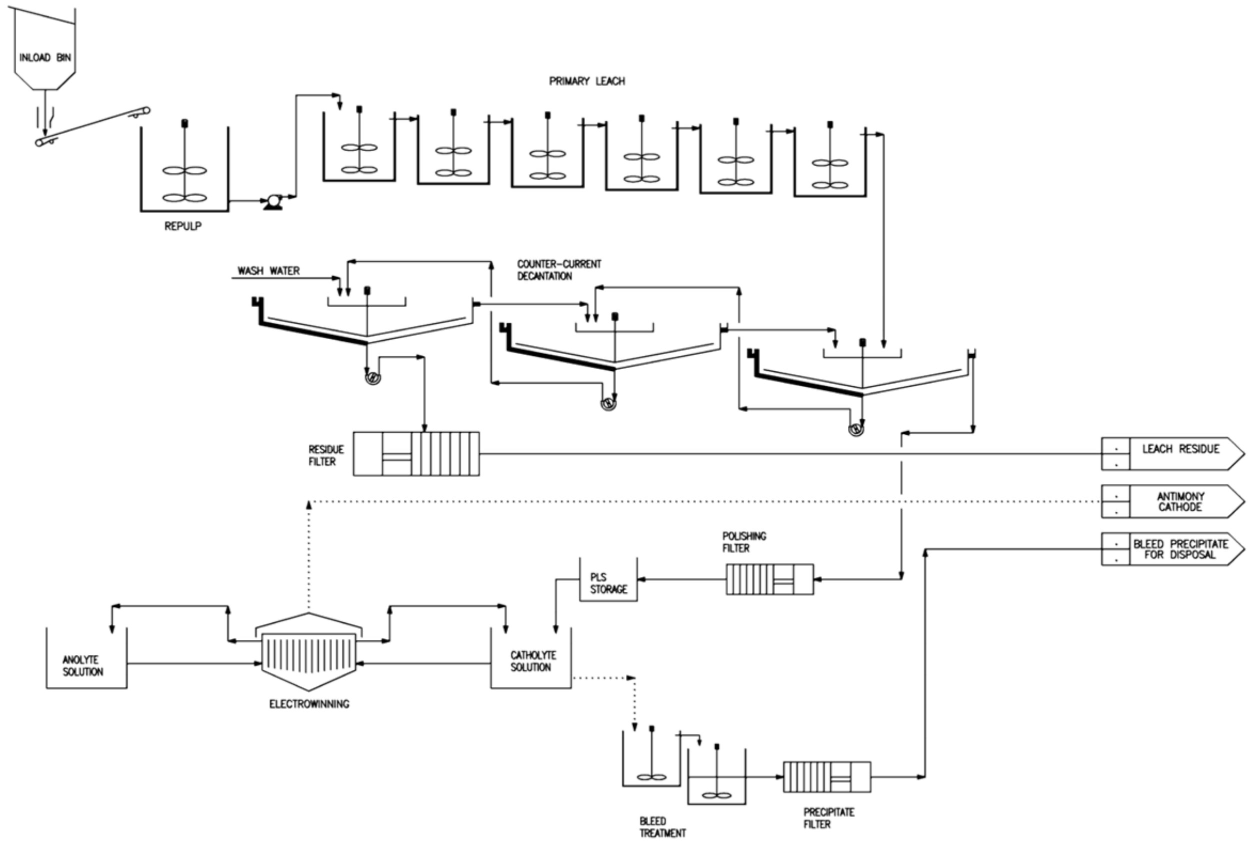
- the payability of the antimony concentrate is low at approximately 65%;
- the antimony mineral resource at the Project is one of the largest known in the western world with significant upside potential;
- potential strategic importance of antimony to US defense and energy sectors considering that there is no domestic production; only recycling and minor treatment of imported concentrates resulting in the US being overly dependent on China for imports;
- recent testwork (summarized in Section 13) has indicated that the potential exists to treat the antimony concentrate using the caustic sulfide leach process previously used at the Sunshine Mine near Kellogg, Idaho to recover ~95% of the antimony as electrowon metal, while returning all of the gold and ~50% of the silver back to the cyanide leach circuit in the process residue.

Given the preceding, Midas Gold commissioned a study by AGP Mining Consultants Inc. (AGP) to estimate the capital and operating costs for a secondary antimony process plant based on the testing presented in Section 13 and the following assumptions:

- 25 st/d nominal plant feed rate;
- concentrate delivered in bulk at 5-15% moisture;
- concentrate grading ~ 50% Sb, by weight;
- regrinding not required prior to concentrate leach;
- camp facilities not required;
- electricity used for steam generation;
- caustic sulfide leach leading to 98% Sb dissolution in less than 2 hours at approximately 90°C;
- electrowinning of antimony directly from the pregnant leach solution;
- recycling of solution back into the leach circuit, with a portion bled off to prevent contaminant buildup;
- leach residue returned to the Stibnite processing plant as bulk filter cake;
- bleed precipitate shipped in bulk bags for disposal;
- overall antimony recovery of 95%;

- the process flow summarized on Figure 24.3; and
- final product cathode shipped on pallets by truck.

Figure 24.3: Antimony Recovery Plant Process Flow Summary Diagram



The total direct CAPEX estimate for the processing plant, based on the preceding assumptions and including mechanical, civil, earthworks, structural, piping, electrical, instrumentation, buildings and mobile equipment costs were estimated at $24.3 million; indirect costs (EPCM, site establishment, first fill, etc.) and contingency (20%) were estimated at $3.7 million and $4.9 million, respectively, bringing the total capital cost estimate to $32.9 million.

The estimated process plant OPEX, which included labor, spares, electricity, reagents, piping, assay laboratory, fuel, consumables and waste disposal, were estimated at $527/st ($581 per tonne) of concentrate processed, or about $0.56 per pound of antimony metal produced.

For an assumed antimony mineral resource of, for example, 200 million lbs, the all-in unit costs, including capital and operating costs, based on the estimates presented in the AGP study yielded an estimated cost of $0.72/lb. For the payability and concentrate transportation costs detailed in Section 19, the deductions associated with antimony concentrate sales are approximately $1.73/lb. Consequently, secondary antimony processing appears to offer a financial advantage over the base case of approximately $1/lb from an undiscounted cash flow perspective based on the analysis and assumptions provided herein. Further, the attractiveness of this alternative would increase significantly if antimony prices were to increase, since the CAPEX and OPEX for the AGP study would remain constant at higher prices, whereas the value of the percentage withheld by the smelters would increase proportionally with the metal price. Therefore, additional metallurgical testing, engineering and cost estimating appear to be warranted, particularly if additional antimony mineral reserves are defined.

Were additional processing of antimony concentrates deemed warranted, this would most likely occur off site; as a result, the current Project design of trucking concentrates offsite would not change.

24.3 PROJECT EXECUTION PLAN

24.3.1 Description

The Project Execution Plan describes, at a high level, how the PFS design presented in this document would be carried out. This plan contains an overall description of what the main work focuses are, Project organization, the estimated schedule, and where important aspects of the design would be carried out.

The Project execution proposed incorporates an integrated strategy for engineering, procurement and construction management (EPCM). The primary objective of the execution methodology is to deliver the Project at the lowest capital cost, on schedule, and consistent with the Project standards for quality, safety, and environmental compliance.

24.3.2 Objectives

The Project execution plan has been established with the following objectives:

- to maintain the highest standard of safety and environmental performance so as to avoid and minimize incidents and accidents;
- to design and construct a process plant, together with the associated infrastructure, that is cost-effective, achieves performance specifications and is built to high quality standards;
- to design and operate the mine using proven methodologies and equipment;
- to optimize the Project schedule to achieve an operating plant in the most efficient and timely manner within the various constraints placed upon the Project; and
- to comply with the requirements of the conditions for the construction and operating license approvals.

24.3.3 Plan of Approach

24.3.3.1 Philosophy

This section describes the execution plan for advancing the Stibnite Gold Project from the current Prefeasibility Design stage to production. The Project execution plan formally identifies and documents the key Project processes and procedures that are required to support the successful execution of the Project including:

- completion of a Feasibility Study;
- develop a Project schedule that encompasses the Feasibility Study through procurement, construction and commissioning;
- consider significant Project logistics;
- develop and implement site communications, construction infrastructure, and water supply for an early and efficient startup;
- plan for early construction mobilization;
- develop practices and protocols that are protective of the environment and ensure compliance with permits and regulations;

- develop an Environmental, Health and Safety Plan that is comprehensive yet concise so that contractors, construction managers, and members of Midas Gold's development team are safe during the field construction phase of the Project;
- develop and execute Project control procedures and processes;
- perform constructability reviews;
- implement Project accounting and cost control best practices;
- issue a cost control plan and a control budget; and
- oversee Project accounting.

Midas Gold would utilize an Engineering, Procurement and Construction Management (EPCM) approach utilizing multiple hard money and low unit cost prime contracts for Construction Management (CM), as the recommended method for executing the Project. The capital cost estimate is based on this methodology. Mine development pre-production work activities as well as the water diversion tunnel, the site access road construction and power transmission line are envisioned to be performed by contractors selected through a pre-qualification and pre-tending process. Because the Project is located in an area with an abundance of qualified contractors, construction would be performed by companies from the Rocky Mountain region, wherever possible. Some items affecting the Project are:

- ability to start work that does not require engineering;
- availability of construction and engineering resources;
- experience of the qualified firms considered and their typical and proposed approach; and
- an approach that utilizes the best resources available (matching contractors to the size of each contract).

As previously mentioned, M3 utilized an EPCM approach as the basis for the capital cost estimate. This approach provides for contracts that would include civil, concrete, structural steel, mechanical, piping, electrical and instrumentation.

The majority of mechanical and electrical equipment required are designed to be procured within North America. Concrete and building construction materials are designed to be sourced locally, wherever possible. Structural and miscellaneous steel, piping, tanks, electrical and miscellaneous process equipment are designed to be sourced within the US, to the extent practical, within the region.

24.3.3.2 Engineering

Engineering is designed to match the plant protocol for drawing titles, equipment numbers and area numbers. Design will continue to produce drawings in the Imperial System of Units (English) format. Drawings and specifications for the PFS have been done in English and are anticipated to remain that way for each subsequent design step.

A site conditions specification would be needed to ensure that vendors are aware of the site conditions. Individual equipment specifications would also be needed.

Engineering control of the PFS design would be maintained through drawing lists, specification lists, equipment lists, pipeline lists, cable schedule, and instrument lists. Control of Engineering Requisitions for Quote (ERFQ) would be performed through an anticipated purchase orders list. Progress would be tracked through the use of the lists mentioned.

As designed, concrete reinforcing steel drawings would be done using customary bar available in the US. Reinforcing bar would be fully detailed to allow either site or shop fabrication.

Structural steel would be detailed using a program such as TEKLA software. Mechanical steel would be dictated utilizing software such as Inventor, TEKLA, or something similar. This would allow fabrication of steel prior to the award of steel installation contracts.

Owner review of engineering progress and design philosophy would of course be an ongoing process.

24.3.3.3 Procurement

Procurement of long delivery equipment and materials is scheduled with their relevant engineering tasks. This would ensure that the applicable vendor information is incorporated into the design drawings and that the equipment would be delivered to site at the appropriate time and supports the overall Project schedule. Particular emphasis would be placed on procuring the material and contract services required to establish the temporary construction infrastructure required for the construction program.

Procurement of major process equipment would be by the EPCM contractor, acting as Agent for Midas Gold through the use of owner-approved purchase order forms. This will include all of the equipment in the equipment list as well as all of the instruments in the instrument list. Some instruments are designed to be part of vendor equipment packages. In addition, structural steel, electrical panels, electrical lighting, major cable quantities, specialty valves and special pipe would also be designed to be vendor packages. Contractors would be responsible for the purchase of common materials only.

Equipment and bulk material Suppliers would be selected via a competitive bidding process. Similarly, construction contractors would be selected through a pre-qualification process followed by a competitive bidding process. It is envisaged that as designed, the Project would employ a combination of lump sum and unit price contracts as appropriate for the level of engineering and scope definition available at the time contract(s) are awarded.

It is intended that equipment would be sourced on a world-wide basis, assessed on the best delivered price and delivery schedule, fit-for-purpose basis.

Equipment would be purchased Free on Board (FOB) at the point of manufacture or nearest shipping port for international shipments. A logistics contractor would be selected to coordinate all shipments of equipment and materials for the Project and arrange for ocean and overland freight to the job site.

The EPCM contractor would be responsible for the receipt of the major equipment and materials at site. The equipment and materials would be turned over to the installation contractor for storage and safe keeping until installed. Bulk piping and electrical materials and some minor equipment would be made part of the construction contracts, and as such would be supplied by the various construction contractors. It is expected that each construction contractor provide for the receipt, storage, and distribution of materials and minor equipment they purchased.

The EPCM contractor would establish a list of recommended pre-qualified vendors for each major item of equipment for approval by Midas Gold. The EPCM contractor will prepare the tender documents, issue the equipment packages for the bid, prepare a technical and commercial evaluation, and issue a letter of recommendation for purchase for approval by Midas Gold. Midas Gold, with the assistance of the EPCM contractor, would conduct the commercial negotiations with the recommended vendor and advise the EPCM contractor of the negotiated terms for preparation of the purchase documents. When approved, the EPCM contractor would issue the purchase order, track the order, and expedite the engineering information and delivery of the equipment to the site.

24.3.3.4 Inspection

The EPCM contractor would be responsible to conduct QA/QC inspections for major equipment during the fabrication process to ensure the quality of manufacture and adherence to specifications. Levels of inspection for major equipment would be identified during the bidding stage, which may range from receipt and review of the manufacturer's quality control procedures to visits to the vendor's shops for inspection and witnessing of shop tests prior to shipment of the equipment. Where possible, inspectors close to the point of fabrication would be contracted to perform this service in order to minimize the travel cost for the Project. Some assistance may also be provided by the EPCM engineering design team.

24.3.3.5 Expediting

The EPCM contractor would also be responsible to expedite the receipt of vendor drawings to support the engineering effort as well as the fabrication and delivery of major equipment to the site. An expediting report would be issued at regular intervals outlining the status of each purchase order in order to alert the Project of any delays in the expected shipping date or issue of critical vendor drawings. Corrective action can then be taken to mitigate any delay.

The logistics contractor would be responsible to coordinate and expedite the equipment and material shipments from point of manufacture to site, including international shipments through customs.

24.3.3.6 Project Services

The EPCM contractor would be responsible for management and control of the various Project activities and ensure that the team has appropriate resources to accomplish Midas Gold's objectives.

24.3.4 Construction

24.3.4.1 Construction Methodology

As currently designed for this prefeasibility report, the construction program is scheduled to start in Year -3. Initial construction work includes clearing and grubbing of the plant site, mass earthwork for site development, Project access road and in-plant roads. Concrete foundations for the process buildings and other support structures are designed to be constructed thereafter. The grinding-flotation building and autoclave buildings are planned to be bridge-frame metal, moment frame structures. The truck shop, the Historic Tailings reclaim building, maintenance shop, and warehouse buildings are currently planned as pre-engineered metal buildings or fabric covered structures. Most of the ancillary buildings on the Stibnite Gold Project site are planned to be modular buildings including the offices, camp, and the ancillary facilities.

As currently designed, construction work is scheduled for approximately 36 months from mobilization to the commencement of commissioning. Earthworks associated with the well field and related facilities would commence after Project permits have been released as soon as a contractor can be mobilized to the field. This work would include completion of the surface diversions, process building foundations and process ponds.

24.3.4.2 Construction Management

Construction Management would likely be done by the EPCM contractor as Agent for the Owner using prime contracts for civil/concrete and structural/mechanical/electrical/piping/instrumentation. The contracting plan is based on utilizing local contractors to execute the construction work packages to minimize mobilization and travel costs. The EPCM contractor would pre-qualify local contractors and prepare tender documents to bid and select the most qualified contractor for the various work packages. Some work packages would include the design, supply, and erection for specific facilities which are specialized in nature. The EPCM team would be comprised of individuals capable of coordinating the construction effort, supervising and inspecting the work, performing field engineering functions, administering contracts, supervising warehouse and material management functions, and performing cost control and schedule control functions. These activities would be under the direction of a resident construction manager and a team of engineers, and locally hired supervisors, and technicians. There would also be a commissioning team to do final checkout of the Project.

Construction progress would be measured by using quantity ledgers for construction quantities to develop percent completion and earned hours by contractors. Quantity surveyors will measure the amount of civil quantities, yards of concrete placed, tons of steel erected, and similar measures for architectural, piping and electrical quantities. Mechanical installations would be measured based on the estimated installation hours from the control estimate developed during detailed engineering.

Some site services would be contracted to third party specialists, working under the direction of the resident construction manager. Construction service contracts identified at this time include field survey and QA/QC testing services.

24.3.5 Contracting Plan

Contracting is an integral function in the Project's overall execution. Contracting for the Stibnite Gold Project per the current PFS design would be done in full accord with the provisions of the Midas Gold/EPCM contract.

A combination of vertical, horizontal, and design-construct contracts may be employed as best suits the work to be performed, degree of engineering and scope definition available at the time of award. The PFS design locates a concrete batch plant on site that is designed to use screened colluvial and alluvial materials native to Meadow Creek or spent ore in the SODA. The design includes a dedicated construction camp at the Stibnite Gold Project site that has been designed to be located approximately one mile from the plant along the upper EFSFSR intersection with the mine access road.

The civil contract would cover all clearing, grubbing, bulk excavation, engineered fill, grading, and possibly, geomembrane lining of the TSF, ponds and pipe trenches.

The concrete contract would include all concrete forming, rebar, placement and stripping. If possible, the batch plant would be tied to the concrete placement contract to leverage the economy of having one management for both functions.

As part of the contracting strategy, a list of proposed contract work packages has been developed to identify items of work anticipated to be assembled into a contract bid package. Depending upon how the Project is ultimately executed and the timing, several work packages may be combined to form one contract bid package. Table 24.4 represents the Proposed Contract Work Package list:

Table 24.4: Proposed Contract Work Package List

No.	Bid Packages:	Comments
1	Materials Testing	Soils, Concrete & Structural Materials
2	Survey	Confirm Existing Terrain. Create Topo of Roadway, Heap Leach & Plant Site Areas
3	Mine Access Road	Includes Roadway Drainage Culverts & Trenching
4	Bridges and Stream Crossings	Multi-plate tunnels
5	Water Diversion Tunnel	Underground mine contractor

No.	Bid Packages:	Comments
6	138 kV Power Transmission Line	Idaho Power to Yellow Pine Substation; a second contractor to erect power transmission line from Yellow Pine Substation to site
7	Construction Camp Installation	Possibly by provider of modular construction camp
8	Main Substation & Oxygen Plant Substation	Includes Emergency Generator Installation & Testing
9	Mine Pre-Stripping Contract	Includes Starter Dam construction
10	Field Electrical Distribution - Sub Station to Process Areas, Camp & Water Pumping	Overhead lines and duct banks from switch gear
11	Water Supply System - Yard Water Piping	Includes Fire Suppression
12	Septic System - Sewer Piping, Plant & Leach Field	Two septic systems required: process plant area and camp area
13	Clearing, Grubbing, Site Excavation, Engineered Backfill, Grading, Trenching, - all Areas	
14	Concrete Work - All Areas	
15	Structural Steel Buildings & Platforms	From foundation bolts. Includes roofing and siding installation.
16	Architectural Finishes	In offices and larger frame structure buildings
17	Field Erected Tanks	Typically part of design-supply-erect contract.
18	Mechanical Equipment	Crusher, conveyors, reclaim feeders, grinding mills, flotation cells, thickeners, pumps, mechanical steel, etc.
19	Process Piping & Field Instrumentation	
20	Instrumentation & Controls Programming	PLC programming, HMI screen development; I/O & communications.
21	Permanent Camp Installation	By camp provider

24.3.6 Project Schedule

A PFS-level schedule has been developed based on the Project description, and objectives philosophy documented herein. The schedule includes Engineering, Contracts, Procurement, Construction, Remaining Site Work, Site Pre-Commissioning, and Site Commissioning activities and is presented on Figure 24.5.

The schedule assumes that pilot plant and feasibility study commence in Year -5 leading into basic and then detailed engineering so that procurement can begin in Q4 of Year -4. Construction would commence shortly thereafter in Q1 of year -3. It is important to note that mine equipment would need to be procured and assembled early starting in Q1 of Year -3 so that pre-stripping could commence in Q2 of Year -2.

The 138 kV power transmission line would also need to start early commencing in Q1 of Year -3 and finishing at the end of Q4 of Year -1.

The Oxygen Plant contract procurement is currently designed to begin in Q1 of Year -3. In order to be able to transport larger items to the project site, the Mine Access Road is schedule so that it would commence in Q1 of Year -3 and continue through Q4 of Year -2.

The autoclave procurement and fabrication commence in Q1 of Year -3 so that they could be delivered, welded into a single shell, stress relieved, pressure tested, and installed by the end of Q2 of Year -1. As currently designed site commissioning could begin shortly thereafter leading to project turnover and the commencement of processing by the end of Year -1.

24.3.6.1 Construction Completion and Handover Procedure

The Construction Completion Procedure is part of the Construction Quality Plan as well as the Project specific Commissioning Plan. Contractors would enter into contractual agreements with Midas Gold to perform certain portions of the work, which includes quality control of their work.

The Commissioning Plan would be designed, developed, and implemented to insure a step-by-step, documented process and procedure for all mechanical, process, electrical/instrumentation completion, checkout and pre-operational testing. Pre-operational testing and commissioning would take place concurrent with mechanical completion. Pre-operational testing, per the current PFS design, is currently scheduled to commence in Q2 of Year -1 and wet commissioning and start-up is scheduled to commence in Q4 of Year -1.

24.3.7 Quality Plan

A Project specific, Quality Plan would be developed and implemented on the site. The Quality Plan would be designed to be a management tool for the EPCM contractor, through the construction contractors, to maintain the quality of construction and installation on every aspect of the Project. The plan, which consists of many different manuals and subcategories, would be developed during the engineering phase and available prior to the start of construction.

24.3.8 Commissioning Plan

The Commissioning Plan would also be designed to be Project specific and characterized as the transition of the constructed facilities from a status of “mechanically” or “substantially” complete to operational as defined by the subsystem list that would need to be developed for the Project. The commissioning group would systemically verify the functionality of plant equipment, piping, electrical power and controls. This test and check phase would be conducted by discrete facility subsystems. The tested subsystems would be combined until the plant is fully functional. Start-up, also a commissioning group responsibility, would progressively move the functional facilities to operational status and performance.

In addition to these activities, the commissioning portion of the work would also include coordination of facilities operations training, maintenance training and turnover of all compiled commissioning documentation in an agreed form.

24.3.9 Environmental, Health and Safety Plan

The Environmental Health and Safety Plan (EHSP) would need to be established for the construction of the Stibnite Gold Project and any other authorized work at the Project site. The EHSP would cover all contractor personnel working on the Project and any other authorized work for the Project.

The EHSP specifies regulatory compliance requirements, training, certifications and medical requirements necessary to complete the Project for all personnel and contractors involved in the Project. The EHSP would include a comprehensive program of sampling and analyses to monitor environmental conditions to insure no negative effects occur during construction. The plan would also include a site-wide Stormwater Management Pollution Prevention Plan (SWPPP) as a preventative measure and a Spill Control and Countermeasures Plan (SPCC). Along with the Operations Procedures, the EHSP would be required to be followed by all Contractor personnel working at the site.

24.3.10 Traffic Management Plan

In order to minimize the disruption along the mine access road and at the mine site, traffic to the site would need to be coordinated by a dispatcher located at the Cascade offsite facility. Midas Gold would adopt a Traffic Management

Plan to guide those travelling between Cascade and the mine site. The plan would be developed in collaboration with the EPCM contractor, construction contractors, suppliers and transportation companies.

24.3.11 Project Organization

Figure 24.4: Project Organization Block Diagram

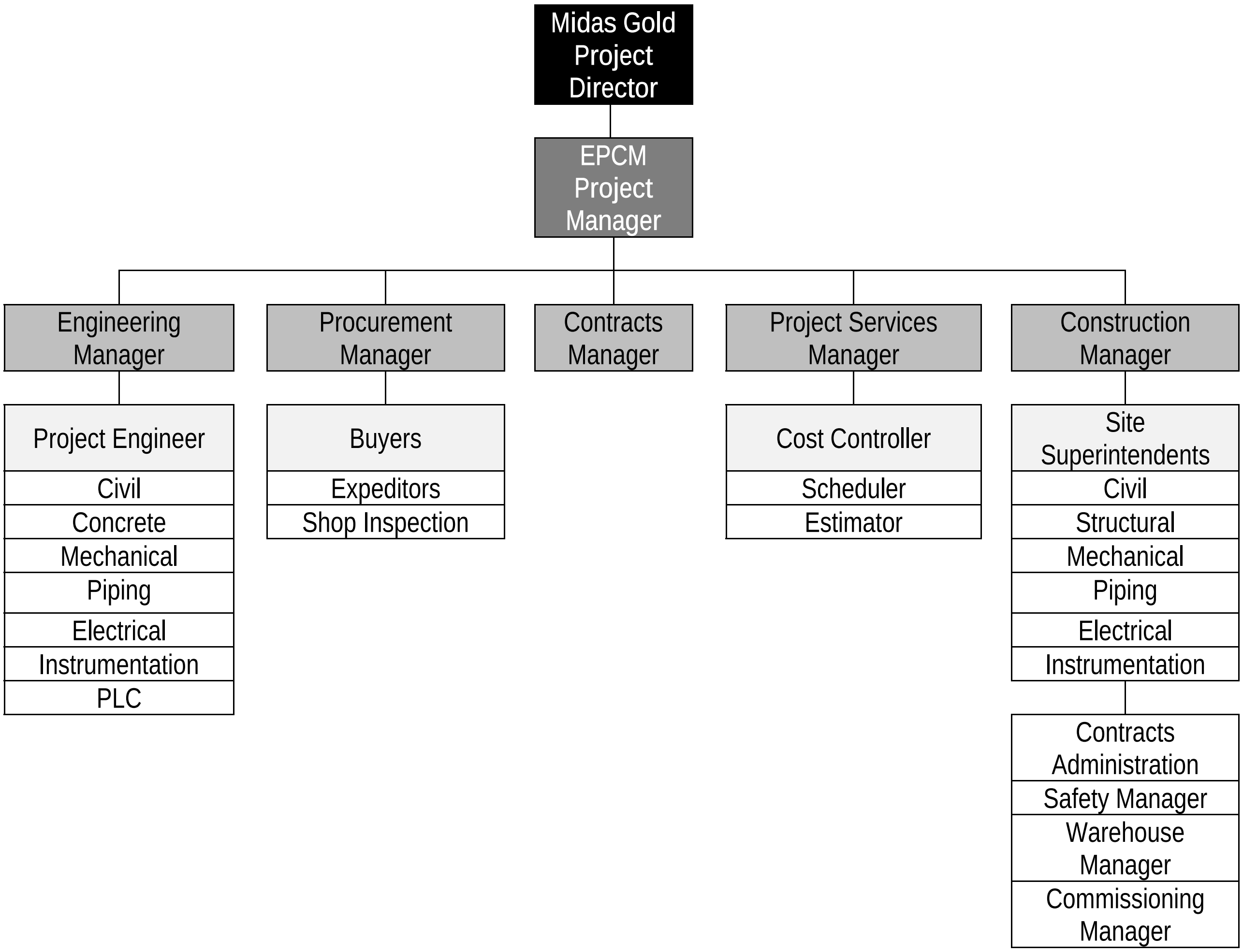


Figure 24.5: Stibnite Gold Project Summary Schedule

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25 INTERPRETATION AND CONCLUSIONS

25.1 INTRODUCTION

According to CIM definition standards for Mineral Resources and Mineral Reserves prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014 (CIM Standards), a Preliminary Feasibility Study is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on the "Modifying Factors" (as defined in the CIM Standards) and the evaluation of any other relevant factors that are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be converted to a Mineral Reserve at the time of reporting. A Preliminary Feasibility Study is at a lower confidence level than a Feasibility Study. Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves; these include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

25.2 INTERPRETATION

The QPs of this Report have reviewed the data for the Stibnite Gold Project and are of the opinion that the Project meets the requirements for a Preliminary Feasibility Study. Opinions from individual QPs on the sections of the PFS that they are responsible for (see Section 2 for responsibilities) are set out in the following subsections.

25.2.1 Surface Rights, Royalties, and Mineral Tenure

Midas Gold is vested with fee simple, mineral, or possessory record title to, or an option to purchase, the Stibnite Gold Project properties described in Section 4 , subject to the royalties, agreements, limitations and encumbrances described in Section 4.

25.2.2 Geology and Mineralization

The understanding of the regional and local geology with regards to the lithology, structure, alteration and mineralization for each of the mineralized zones and deposit types discussed in Sections 7 and 8 is sufficient to estimate the Mineral Resources and Mineral Reserves contained herein.

25.2.3 Exploration

The previous drilling exploration programs, along with the geologic mapping, geochemical and geophysical studies, and petrology and mineralogy research carried out to date, reasonably supports the potential for expansion of defined deposits, potential for discovery of high-grade underground mineable prospects, and the potential for discovery of new bulk mineable prospects as discussed in Section 9 .

25.2.4 Drilling and Sampling

The drilling methods, recovery, collar survey, downhole survey, and material handling for the samples used in the Mineral Resource and Mineral Reserve estimates for this Report are sufficient to support the Mineral Resource and Mineral Reserve estimates contained in this Report, subject to the assumptions and qualifications contained in Sections 10 and 11 .

25.2.5 Data Verification

The data used for estimating the Mineral Resources for the Hanger Flats, West End, Yellow Pine and Historic Tailings is adequate for the purposes of this Report and may be relied upon to report Mineral Resources and Mineral Reserves based on the conditions and limitations set out in Section 12.

25.2.6 Metallurgy

The metallurgical testing conducted on samples from West End, Hangar Flats, Yellow Pine, and the Historic Tailings included extensive mineralogical studies and developmental metallurgical testing on various ore types from each of the deposits. The developmental metallurgical testing and analysis detailed in Section 13 supports the selection of the process flow sheet that proved successful when applied to each of the deposits, making it possible to design a single plant that can process all ores from the Project as they are mined subject to the conditions and limitations set out in Section 13 .

25.2.7 Mineral Resources

The Mineral Resource estimates in Section 14 are accurate to within the level of estimate required for categorization as Inferred and Indicated Mineral Resources with the latter suitable for use in a Preliminary Feasibility Study, subject to the conditions and limitations set out in Section 14, and these estimates were performed consistent with industry best practices and demonstrate reasonable prospects for economic extraction, as required by NI 43-101.

25.2.8 Mineral Reserves

A thorough review of the designs, schedules, risks, and constraints of the Project detailed within this Report and given that there is, in the opinion of the QP responsible for Section 15, a basis for an economically viable Project after taking into account mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social, governmental factors and other such modifying factors, thereby supporting the declaration of Mineral Reserves. Subject to the conditions and limitations contained in this Report, this PFS demonstrates that, as of the date of this Report, extraction can reasonably be justified. The term 'Mineral Reserve' does not necessarily signify that all governmental approvals have been received; it does signify that there are reasonable expectations that such approvals will be granted.

25.2.9 Mine Plan and Schedule

The mine plan and schedule detailed in Section 16 have been developed to maximize mining efficiencies, while utilizing the current level of geotechnical, hydrological, mining and processing information available and are, subject to the conditions and limitations set out in Section 16, sufficient to support the declaration of Mineral Reserves.

25.2.10 Metallurgical Recovery

The recovery methods including the major unit operations detailed in Section 17 comprising, primary crushing, SAG and ball mill grinding, antimony flotation (when warranted), bulk auriferous sulfide flotation, auriferous sulfide concentrate pressure oxidation, cyanidation of the pressure oxidation residue, CIL processing of the flotation tailings, precious metal recovery to doré and tailings detoxification are sufficient to demonstrate recoveries to support the mine planning and economics detailed herein, and the declaration of Mineral Reserves.

25.2.11 Infrastructure

The on-site and off-site infrastructure detailed in Section 18 is designed and cost estimated to a level of detail that supports Project viability and the economics detailed herein.

25.2.12 Market Studies and Contracts

The doré and antimony concentrate market studies detailed in Section 19 are consistent with industry standards and market patterns, and are similar to contracts found throughout the world. The metal prices selected for the four economic cases in this Report represent a probable range of scenarios that support a prefeasibility economic analysis.

25.2.13 Environment, Permits, and Social and Community Impacts

Section 20 summarizes the reasonable available information on: environmental studies conducted to date and the related known environmental issues associated with the Project, the Project related social and community impacts and benefits, the remediation of legacy impacts built into the design for and execution of the Project, the Project permitting requirements, and the requirements and plans for waste rock and tailings storage. Additionally, mine closure, reclamation and mitigation are discussed and cost estimated to a level of detail that supports Project economic and technical viability to the level of a Prefeasibility Study and the economics detailed herein.

25.2.14 Capital and Operating Costs

The capital and operating costs detailed in Section 21, which were derived from several previous sections of the Report are, subject to the conditions and limitations in this Report, designed and cost-estimated to a level of detail that supports Project economic and technical viability to the level of a Prefeasibility Study and the economics detailed herein.

25.2.15 Financial Analysis

The financial analysis presented in Section 22 illustrates that the Project economics, subject to the conditions and limitations in this Report, are positive and can support estimation of Mineral Reserves and the demonstration of technical and economic viability to the level of a Prefeasibility Study.

25.3 CONCLUSIONS

The financial analysis presented in Section 22 demonstrates that the Stibnite Gold Project is technically viable and has the potential to generate robust economic returns based on the assumptions and conditions set out in this Report and this conclusion warrants continued work to advance the Project to the next level of study, which is a Feasibility Study.

The QPs of this Report are not aware of any unusual, significant risks or uncertainties that could be expected to affect the reliability or confidence in the Project based on the data and information available to date.

25.4 RISKS

As with most projects at the preliminary feasibility level, there continues to be risks that could affect the economic potential of the Project. Many of the risks relate to the need for additional field information, laboratory testing, or engineering to confirm the assumptions and parameters used in this Report. External risks are, to a certain extent, beyond the control of Midas Gold and are much more difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. Table 25.1 identifies what are currently deemed to be the most significant internal Project risks, potential impacts, and possible mitigation approaches. In summary, the Project-specific risks identified following the PFS include:

- Mineral Resource modelling;
- geotechnical engineering;

- loss of gold into Sb concentrate;
- metallurgical recoveries;
- water management;
- water geochemistry; and
- development or construction schedule.

Table 25.1 Project Risks Identified Following the PFS

Risk		Explanation / Potential Impact	Possible Risk Mitigation
General Risks Common to the Mining Industry			
GR1	CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of Project success. An increase in OPEX of 20% would reduce the after tax $NPV_{5\%}$ by approximately $20 M using current open pit designs. If OPEX increases, then the mining cut-off grade would increase and, all else being equal, the size of the optimized pit would reduce, yielding fewer mineable tons and less recoverable gold.	Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost-reduction measures would assist in the accuracy of cost estimates.
GR2	Permit Acquisition or Delay	The ability to secure all of the permits to build and operate the Project is of paramount importance. Failure to secure the necessary permits could stop or delay the Project.	A thorough Environmental and Social Impact Assessment of a Project design that gives appropriate consideration to the environment and local community expectations and input is required.
GR3	Ability to Attract Experienced Professionals	The ability of Midas Gold to attract and retain competent, experienced professionals is a key success factor for the Project. High turnover or the lack of appropriate technical and management staff at the Project could result in difficulties meeting Project goals.	The early search for, and retention of, professionals may help identify and attract critical people.
GR4	Falling Metal Prices	A drop in metal prices during the mine development process could have a negative impact on the profitability of the operation, especially in the critical first years.	Begin construction when the outlook is good for price improvement and have mitigating strategies, such as hedging or purchase of puts, and supporting analyses to address the risk of a downturn.
GR5	Change in Permit Standards, Processes, or Regulations	A change in standards, processes, or regulations could have a significant impact on project schedules, operating cost and capital cost.	Participate in legislative and regulatory processes to ensure standards remain protective, fair and achievable.
Stibnite Gold Project Specific Risks			
PR1	Mineral Resource Modelling	Certain Mineral Resources were estimated with data that included historic samples and these may have not had sufficient confirmation from modern drilling and sampling to support a production decision, which introduces some level of risk and uncertainty. The risk is the level of certainty in the Mineral Resource estimates and whether they can be confirmed with additional drilling.	Further confirmation drilling and verification are needed to remove, replace or supplement historic sample-based data from Mineral Resource estimates, especially in the Yellow Pine deposit.

Risk		Explanation / Potential Impact	Possible Risk Mitigation
PR2	Geotechnical Engineering	The geotechnical condition of the soils under the WRSF, plant, and infrastructure facilities may be different than assumed and could have financial implications on the Project CAPEX and/or OPEX.	Further field investigations are required to support the Feasibility Study.
PR3		The geotechnical nature of the open pit wall rock, including the nature of faults and secondary geological structures, could impact the allowable pit slopes, which could impact mineable tons, strip ratio and overall Project economics negatively or positively.	
PR4	Loss of Gold into Sb Concentrate	The flotation circuit design is based on sequential flotation (the flotation and removal of antimony sulfides followed by the flotation of gold sulfides from the antimony flotation tailings) for a portion of the mill feed. Higher than expected gold losses in the antimony flotation circuit would negatively impact the viability of the antimony circuit.	Additional metallurgical work (i.e. pilot plant testing) should be completed as part of the Feasibility Study, which should increase confidence in the projected metallurgical recoveries.
PR5	Metallurgical Recoveries	Changes to metallurgical assumptions could lead to reduced metal recovery and revenue, increased processing costs, and/or changes to the processing circuit design, which would all negatively impact the Project economics. A 1% reduction in total gold recovery would reduce the Case B $NPV_{5\%}$ by about $29M.	Pilot plant runs with appreciable larger samples should be completed to support the Feasibility Study, to increase the confidence of the recovery assumptions and overall process design.
PR6	Water Management	Water management is a critical component of the Project. While a comprehensive site-wide water balance model and 3D groundwater model was used to design the ground and surface water diversion and interception systems, more field information is required to improve the accuracy of the water balance, size diversion channels and settling ponds, design treatment facilities, and develop comprehensive long-term closure designs.	Complete additional hydrogeological fieldwork in the Meadow Creek Valley to improve the understanding of the groundwater regime around the Hangar Flats open pit. Continue to collect and analyze on-site groundwater, surface water, and meteorological data to enhance hydrological knowledge of the site.
PR7	Water Geochemistry	Based on test work completed, it was assumed that waste rock storage facilities (WRSFs) do not need to be lined and that metal leaching (ML) and acid rock drainage (ARD) would be manageable and achieve regulatory water discharge limits without treatment. If future ML/ARD testing indicates that the WRSFs require lining or special treatment, then CAPEX and/or OPEX would increase.	Additional geochemical testing and modeling should be completed to further refine the appropriate water management strategies for the Project.

Risk		Explanation / Potential Impact	Possible Risk Mitigation
PR8		While current test work indicates otherwise, comingling of tailings streams may lead to water chemistries of reclaimed waters that adversely affect flotation metallurgy.	More refined geochemical testing of the tailings supernatant should be completed in the next level of study. More makeup water or alternative water management plans may need to be investigated, depending on the outcome.
PR9		Long-term, post-closure, pit lakes in the Hangar Flats and West End open pits were assumed to be of acceptable discharge quality after a period; were this not the case, additional treatment costs may be incurred.	Additional testing and pit lake modeling will be required in the next phase of study to verify this assumption.
PR10	Development or Construction Schedule	The Project development could be delayed or extended for a number of reasons, which would impact Project economics.	If an aggressive schedule is to be followed, FS field work and critical path laboratory testing should begin as soon as possible.

25.5 OPPORTUNITIES

There are many significant opportunities that could improve the economics, and/or permitting schedule of the Project beyond those common to the sector (such as increasing metal prices, falling input costs, etc.). The major opportunities that have been identified at this time are summarized in Table 25.2. Further information and assessments are needed before these opportunities could be included in the Project economics.

The opportunities are separated into general opportunities common to the mining industry, and Project-specific opportunities unique to the Stibnite Gold Project. The Project-specific opportunities are further categorized into three broad categories of potential to improve the Project Net Present Value ($NPV_{5\%}$); the categories, and a brief listing the opportunities, are provided below:

- High potential benefit opportunities (potential to increase $NPV_{5\%}$ by more than $100 million) include:
  - in pit conversion of Inferred Mineral Resources to Mineral Reserves;
  - out of pit conversion of Inferred Mineral Resources to Mineral Reserves adjacent to the current Mineral Reserves;
  - in pit conversion of unclassified material currently treated as waste rock to Mineral Reserves;
  - improved continuity of higher grade gold mineralization in the Yellow Pine Pit, particularly around the area influenced by excluded or limited Bradley drilling could enhance gold grades in these areas, which are scheduled early in the Project life;
  - increased Mineral Resources and Mineral Reserves in West End by adding fire assay information in areas where only cyanide assays were available. Could also potentially increase grade.
  - potential additional antimony mineralization in areas where Bradley data was eliminated (which are scheduled early in the Project life) and/or areas where antimony was not assayed;
  - potential for an underground Mineral Reserve at Scout that would likely be antimony rich;
  - potential for an underground Mineral Reserve at Garnet that has the potential to be relatively high grade; and
  - exploration potential for new deposits.
- Medium potential benefit opportunity (potential to increase $NPV_{5\%}$ by $10 to $100 million) include:
  - Metallurgical improvements that improve the Project economics;
  - Secondary antimony processing to enhance payability;
  - Potential definition of antimony as a critical mineral in US legislation;
  - Open pit slope steepening through collection of additional geotechnical information and analysis;
  - Onsite quicklime generation; and
  - Alternative funding sources for off-site infrastructure; and
  - Utilizing preowned equipment to reduce CAPEX and development timelines.
- Low Potential Benefit Opportunity (potential to increase $NPV_{5\%}$ by less than $10 million include:
  - Tungsten recovery as a by-product;

Using the antimony credit in open pit optimization, increasing Mineral Reserves.

Table 25.2 Project Opportunities Identified Following the PFS

Opportunity		Explanation	Potential Benefit
General Opportunities Common to the Mining Industry			
GO1	Permit Acquisition	In the same way that permit acquisition is a potential risk to the Project schedule, it may also be an opportunity. Idaho is characterized as having a low jurisdictional risk, and as a mining friendly state. In addition, the brownfields nature of the Project site may provide a significant impetus to see the Project, with the extensive remediation of legacy impacts built into the design, accelerated.	The opportunity to shorten the permitting schedule exists, bringing value forward.
GO2	Rising Metal Prices	Increases in metal prices, especially gold, would increase revenue and Project economics.	Increased revenue enhances financial factors.
GO3	Reagent/Fuel Price Decreases	Reduction in reagent and consumable prices, especially lime, fuel, and oxygen, has the potential to decrease operating costs and enhance the Project economics.	Lower OPEX may lead to higher net revenue and enhanced Project economics.
Project Specific Opportunities with High Potential Benefit			
PO1	In-pit conversion of Inferred Mineral Resources to the Indicated category	Significant Inferred Mineral Resources exist in each of the Project deposits, including material within the Mineral Reserve pits; these Mineral Resources are currently treated as waste rock and therefore a cost. Conversion of Inferred Mineral Resources within the Mineral Reserve pits to the Measured and Indicated Mineral Resources categories would increase Mineral Reserves, reduce strip ratios and improve overall Project economics.	A tabulation of the Inferred Mineral Resources within the PFS pits, using a Net of Process cutoff of $0.001/ton, results in contained mineralization above cutoff of 10.8 million tons containing approximately 347 koz Au, 524 koz Ag, and 9,544 klbs Sb at average grades of 0.032 oz/st Au (1.1 g/t), 0.049 oz/st Ag (1.7 g/t), including 878 thousand tons containing 0.5% Sb. 100% conversion of this mineralization to Mineral Reserves would reduce the Project strip ratio from 3.5:1 to 3.1:1.
PO2	Out of pit conversion of Inferred Mineral Resources to the Indicated category	Additional drilling in the vicinity of the three Project pits has the potential of increasing the grade and tonnage of the Mineral Reserves by (a) converting above cutoff Inferred Mineral Resources to Indicated, (b) supporting expanded pits that bring current above cutoff Indicated Mineral Resources outside the pits into Mineral Reserves and (c) adding new above cutoff mineralization in currently poorly drilled areas.	Increases in Mineral Reserve tonnages, especially at higher grades, could improve the Project economics, especially if those improvements could be realized in the early stages of development.

	Opportunity	Explanation	Potential Benefit
PO3	In Pit Waste Rock conversion to Mineral Resource	Significant volumes within each of the Mineral Reserve pits are comprised of unclassified material based on a lack of drilling. As drilling continues within the pit limits, some portion of this material could be converted to Mineral Resources above cut-off, increasing Mineral Reserves and reducing the strip ratios.	Increases in Mineral Reserves and reductions in strip ratio would lead to a longer Project life and potentially reduced CAPEX and OPEX. This opportunity is particularly evident at Hangar Flats, where a significant proportion of the in pit material classified as waste rock comprises unclassified blocks due to a lack of drill data west of the MCFZ.
PO4	Improve the continuity of mineralization in the Yellow Pine Pit	As discussed in Section 12, a significant amount of historic information was excluded from Mineral Resource estimation, including holes missing critical supporting information in the core of the Yellow Pine Mineral Resource. Specifically, in the case of the antimony Mineral Resource in Yellow Pine, Bradley Mining Company samples were excluded due to apparent high bias with respect to antimony grade, even though some of these holes were focused within the highest grades portions of the antimony mineralization. As a result, the antimony Mineral Resource may be understated in both tons and grade.	Further drilling in the core of the Yellow Pine Mineral Resource, where known mineralization occurs but data was not used in the current Mineral Resource estimates, could demonstrate continuity helping to increase grade in areas or convert material from waste rock to Mineral Reserves. Inclusion of the historic data would have increased estimated contained gold in Yellow Pine by approximately 4% (approximately 180k oz) and contained antimony by up to 20% (approximately 18 million lbs) as compared to the Mineral Resource estimates stated herein.
PO5	Increase in Mineral Resources and Reserves in West End from CN Assay	Partial or spot Au FA is prevalent throughout the West End deposit, where available AuCN assays do not adequately define the transition from oxide to sulfide gold and likely significantly underestimate the contained gold in the transition and sulfide portions of the deposit. As discussed in Section 14, this issue was addressed by removing 70 drill holes with incomplete AuFA assays from the current Mineral Resource estimate, likely resulting in an underestimation of the total gold grade and quantum of the Mineral Resources.	Additional drilling in the areas where AuCN assays have confirmed and potentially under-predicted the existence of gold mineralization could increase the quantity and grade of the Mineral Resources and increase the Mineral Reserves and reduce the strip ratio in the West End open pit. Compared to the PEA Mineral Resource estimate, approximately 300 koz of gold were eliminated through the removal of this data and other contributing factors.
PO6	Potential Additional Antimony	In the Mineral Resource estimates used in this Report (see Section 14), some of the pre-Midas Gold data used did not include assays for antimony and thus was not available for inclusion in the Mineral Resource estimates, potentially resulting in an underestimation of the actual antimony grades.	Additional drilling in the areas that lack antimony assays could increase the grade and quantity of the antimony Mineral Resources in the Yellow Pine and Hangar Flats deposits.

	Opportunity	Explanation	Potential Benefit
PO7	Potential for Scout Underground Mineral Reserve	Scout is a potentially underground mineable Au-Ag-Sb exploration prospect (see Section 9). It has been identified as a conceptual potential underground geologic potential in the range of 2-5 million tons containing between 50 - 300 koz Au; 40 -150 Mlbs Sb; and 300 - 1,500 koz Ag with target dimensions of approximately 25 - 75 ft thick (true), 2,000 - 3,000 ft along strike and extending 250 - 300 ft down dip at grades ranging from 0.03 - 0.06 oz/st Au (1 - 2 g/t), 1 - 4% Sb, and 0.15 – 0.30 oz/st Ag (5 - 25 g/t).	Addition of a high-grade underground Mineral Reserve at Scout could potentially enhance Project economics by blending in a percentage of high grade feed early in the Project life and would help to smooth and extend the antimony concentrate production profile.
PO8	Potential for Garnet Underground Mineral Reserve	The dimensions of mineralized material located beneath and beyond the boundaries of the former Garnet open pit, as determined by simple polygonal estimation methods from historic drilling and geophysical data, outlines a conceptual underground geologic potential in the 1 - 2 million ton range containing 250 – 500 koz Au approximately 30 - 60 ft thick (true) by 160 – 250 ft wide by 1,300 - 1,800 ft long down plunge at grades ranging from 0.15 – 0.23 oz/st Au (5 - 8 g/t).	Addition of a high-grade underground Mineral Reserve at Garnet could potentially enhance Project economics by blending in a percentage of high grade feed early in the Project life, increasing annual gold production.
PO9	Exploration Potential for Additional Deposits	As discussed in Section 9, the expansion of known Mineral Resources and the addition of new deposits may be possible with further drilling. Based on preliminary geophysical results, the Project area has several exploration targets that justify drilling and may or may not lead to the discovery of additional underground and/or open pit deposits.	The expansion of the Project's Mineral Resources could potentially lead to a longer Project life and/or greater operating flexibility and potentially the justification for a higher throughput. This becomes particularly important, as demonstrated by the economic margin from Yellow Pine vs. Hangar Flats or West End, if higher-grade Mineral Resources are defined that defer lower-grade Mineral Resources currently utilized in the economic analysis.
Project Specific Opportunities with Medium Potential Benefit			
PO10	Metallurgical improvements that improve the Project economics	Several metallurgical opportunities exist, but require confirmation testing. The principal testing required to potentially improve Project metallurgy include: grindability studies, mineralogical profiling, flow sheet upside investigations, follow-up bench scale flotation and tailings leach studies, further development of antimony concentrate processing, a gold pilot plant, an antimony pilot plant, further whole ore and POX residue leach development testing to include carbon adsorption isotherm testing, and inclusion of silver data in all testing to further define recoveries.	Further metallurgical testing is needed better define these opportunities and their impacts.

	Opportunity	Explanation	Potential Benefit
PO11	Secondary antimony processing	Secondary antimony processing of the antimony concentrates to produce a marketable antimony product such as: antimony trioxide, antimony metal, sodium antimonite, or other, has been tested on a preliminary basis with positive results (See Section 13) and could result in enhanced Project economics. These benefits increase as antimony prices increase due to the percentage payability for antimony concentrates vs, stable costs for secondary processing. In addition, secondary antimony processing would largely eliminate any risk related to gold lost to antimony concentrates during flotation, since most of such gold could be recovered from leach residues after secondary antimony processing.	Secondary antimony processing would allow a significant portion of antimony products to be produced in the USA, reduce US reliance on offshore suppliers, as well as improve terms for payable metal. Additionally, in the current flow sheet, antimony flotation is performed prior to gold flotation and the antimony concentrate is shipped offsite for further processing. As a result, any gold lost to the antimony flotation circuit is also shipped offsite, resulting in the loss of gold or reduced payability. Secondary antimony processing at a nearby plant could allow the gold lost in the concentrate to be fed back into the POX Circuit, post-antimony processing, to recover some of the gold lost to the antimony concentrate.
PO12	Potential definition of antimony as a critical mineral	S.1113 Critical Minerals Policy Act and HR 4402 Critical Minerals Bill are not expected to become law during the current Administration. However, similar legislation is expected to be reintroduced in 2015. Such bills are intended to improve permitting predictability and timelines.	Passage of such legislation could reduce the timeline for environmental assessment and permit acquisition for eventual development
PO13	Open pit slope steepening	The open pit slope designs for the PFS were based off of a PFS-level field and laboratory test program; however, based on existing pit slopes at the site, there is some indication that slopes could be steepened, subject to the results of additional geotechnical drilling and analyses.	An increase in overall pit slopes has the potential to add gold to the in-pit tonnage as Mineral Resources below the current Mineral Reserve pits could come in to the Mineral Reserve pit and/or reduce the overall strip ratio and waste tonnage mined.
PO14	Onsite quicklime generation	The Project area has known limestone occurrences that may be suitable for developing quicklime that could be used in mineral processing.	Quicklime is the highest Project reagent cost at over $16M annual average OPEX LOM; the PFS was developed assuming that quicklime would be purchased and trucked into the site. The ability to make the quicklime on site could significantly reduce quicklime costs and significantly reduce the Project OPEX.
PO15	Preowned Equipment	If available at the time of construction decision, some major capital equipment components may be available as pre-owned items suitable for the Project, with some modifications to the equipment and/or Project.	If acquired on favorable terms, could reduce capital costs and lead times.

	Opportunity	Explanation	Potential Benefit
PO16	Alternative funding for off-site infrastructure	Government funding programs such as the Transportation Investment Generating Economic Recovery, or TIGER Discretionary Grant program, provides a unique opportunity for the DOT to invest in road, rail, transit and port projects that promise to achieve critical national objectives. Since 2009, Congress has dedicated more than $4.1 billion to fund projects that have a significant impact on the Nation, a region or a metropolitan area. Similarly, P3 (public-private partnerships) have been used for infrastructure development when the benefits extend to the broader community.	Alternative funding could move costs out of CAPEX and/or OPEX.
Project Specific Opportunities with Low Potential Benefit			
PO17	Tungsten contribution	The YP open pit was mined in the early 1940s for its tungsten; the pit was the largest single source of tungsten for the WWII Allied war effort. Tungsten content remaining in the YP and HF deposits is unknown due to limited assay data and highly variable distribution.	The addition of a tungsten component to the overall value of the Project cannot be quantified until Mineral Resources are defined or production commences and sufficient tungsten is identified in the production stream, but there remains a possibility that tungsten could contribute to the Project economics on an incremental basis.
PO18	Conversion of additional legacy waste materials to ore	There are several million tons of historical waste stored at Yellow Pine and West End and on the Hecla heap that limited data suggests some may be above cut-off grade, which can only be determined through additional drilling. This material is currently treated as waste and therefore a cost center in the PFS.	If sufficient tonnage and grade is defined, this material could be reprocessed, generating additional revenues and reducing strip ratios.
PO19	Antimony credit in open pit optimization	The contribution of antimony sales was not taken into account for the open pit optimization work; only gold values were used. The inclusion of antimony revenue would lower the gold equivalent cut-off grade and likely increase the mineable portion of the Mineral Resource.	The Project economics and life may be enhanced with the inclusion of antimony revenue in the open pit optimization.

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26 RECOMMENDATIONS

Based on the results of this Preliminary Feasibility Study, it is recommended that the Project move forward to the next phase. A detailed list of recommendations and work programs has been developed, including estimated costs, that would move the Project through to completion of a Feasibility Study (FS) and, if warranted, through the regulatory process for mine development. Total estimated cost for completion of this phase is $22.3 million. An additional $22.5 million is identified as discretionary expenditures that would target certain opportunities identified in Section 25 that could enhance the PFS case but that are not required to complete a FS or for permitting. The estimates have been factored on the basis of some success in each of these areas; were poor results to be received early in the evaluation of the opportunity, discretionary expenditures for this activity would be significantly less than indicated, while exceptional success or exceptional results in a particular area of activity could require higher expenditures than indicated. In addition, it is not likely that all discretionary activities would be undertaken in the timeframe leading up to the completion of the FS; some, such as drilling the CN assay areas at West End, may wait for some time post-production due to the current schedule for their extraction in the LOM plan unless gold prices warrant an expedited approach.

The detailed recommendations have been grouped into logical discipline categories including:

- Mineral Resource evaluation and exploration;
- Field programs required for FS;
- Metallurgical testing required for FS;
- Project optimization and FS engineering; and
- Environmental, regulatory affairs and compliance.

Some recommendations are fundamental to moving the Project forward, whereas other items are discretionary. Table 26.1 summarizes the recommendations and work programs, and separates the costs associated with the work program into core and discretionary categories.

Table 26.1: Project Recommendations, Work Program and Budget

Recommendations and Work Program		Unit	Quantity	Estimated Costs ($000s)	
				Core	Discretionary
Mineral Resource Evaluation and Exploration					
R1	Further replacement and/or confirmation of pre-Midas Gold drilling, especially at Yellow Pine, to improve confidence, continuity, and potentially grade/contained metal, especially for Sb.	feet of drilling	16,400	3,600	-
R2	Selective, high-value drilling that targets converting in-pit Inferred Mineral Resource to Mineral Reserves, increasing grade and/or reducing strip ratio in all pits.	feet of drilling	18,000	-	3,600
R3	Selective, high value drilling targeting near-pit opportunities for additional Mineral Reserves, especially at Yellow Pine.	feet of drilling	10,000	-	2,000
R4	Selective testing of in-pit unclassified material for potential additional Mineral Reserves and lower strip ratio for all pits, but especially at Hangar Flats west of the MCFZ.	feet of drilling	10,000	-	2,000
R5	Additional drilling at West End to determine total gold, in areas where only AuCN assay data is available, for potential higher grades, additional Mineral Reserves and/or lower strip ratio.	feet of drilling	13,800	-	3,000
R6	Definition of small tonnage, high grade Mineral Resources at Garnet, Upper Midnight, and Scout for potential high margin mill feed that could supplement early production.	feet of drilling	30,000	-	6,600

	Recommendations and Work Program	Unit	Quantity	Estimated Costs ($000s)	
				Core	Discretionary
R7	Continued exploration including mapping, geochemical sampling, and potentially drilling geared toward defining additional Mineral Resources.	feet of drilling	16,400	-	4,000
R8	Increase geologic understanding and control at known deposits to improve Mineral Resource interpretation and geo-metallurgy.	each	1	100	-
Field Programs Required for FS					
R9	Geotechnical drilling and testing at HF and WE to support the FS open pit designs (some overlap with Mineral Resource expansion drilling), potentially increasing pit slopes, increasing Mineral Reserves as pits deepen into below pit Mineral Reserves and reducing strip ratios.	feet of drilling	6,000	1,000	-
R10	Penetration testing within TSF dam footprint to support the FS engineering design.	each	1	100	-
R11	Shallow sampling of alluvium via test pits or hand-held augur drilling in TSF footprint to characterize liner bedding and borrow materials.	each	1	100	-
R12	Shallow sampling of alluvium via test pits or hand-held augur drilling and bedrock to define concrete aggregate borrow sources.	each	1	100	-
R13	Geotechnical drilling at process plant and truck shop areas for FS foundation designs.	feet of drilling	3,000	500	-
R14	Installation of large-diameter well and pumping test near HF open pit to support FS water supply and HF open pit dewatering system designs, and for closure-related pit lake modeling.	each	1	100	-
Metallurgical Testing Required for FS					
R15	Additional metallurgical testing to optimize grinding, recoveries, reagent consumption and other operating parameters.	each	1	800	300
R16	Complete additional testing of secondary antimony processing to determine if circuit should be included in FS.	each	1	200	500
R17	Complete gold flotation and pressure oxidation pilot plant to better define operating parameters, reagent consumptions, metallurgical recoveries and environmental performance parameters.	each	1	1,300	500
R18	Complete additional metallurgical testing to improve understanding of, and potentially optimize, silver recoveries.	each	1	100	-
Feasibility-Level Engineering					
R19	Complete a multi-discipline, Project-wide enterprise optimization study using cost and technical information developed for the PFS to develop a roadmap for the FS.	each	1	500	-
R20	Concurrent to permitting, complete feasibility-level engineering and design based on additional information gathered post-PFS.	each	1	3,000	-
Environmental, Regulatory Affairs and Compliance					
R21	Advance environmental and closure-related technical studies based on additional field and laboratory information generated.	each	1	700	-
R22	Continue baseline data collection, environmental compliance and reclamation.	each	1	3,900	-
R23	Continue to advance regulatory process including Plan of Operations, Federal EIS under NEPA, and Federal and State permits.	each	1	6,200	-
Totals				22,300	22,500

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APPENDIX I: PREFEASIBILITY STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

CERTIFICATE of QUALIFIED PERSON

I, Lee A. Becker, P.E., P.Eng. do hereby certify that:

1. I am currently employed as Director of Project Services and Controls and Project Manager by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, Ste. 101
   Tucson, Arizona 85704, U.S.A.

2. I graduated with a with a degree in Bachelor's of Science in Energy Engineering from the University of Arizona in 1986.
3. I am a registered professional engineer in good standing in the State of Arizona in the area of mechanical engineering (No.32652).
   - I am also registered as a professional engineer in the states of California (No.32035), Nevada (No.015302), Washington (No.28461), and Alaska (No.9182).
   - In addition, I am a registered professional engineer in Yukon Canada (No.2349), and Ontario Canada (No.100513192).
4. I have worked as an engineer for a total of 33 years since my graduation from the University of Arizona. My experience includes 24 years at M3 Engineering & Technology Corporation working on all aspects of mine plant development for base and precious metals projects specifically plant layout, infrastructure, scheduling and cost estimation. As Project Manager, I have been involved with studies as well as full engineering, procurement, and construction management (EPCM) projects.
5. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
7. I am responsible for Sections 18 (except 18.2, 18.9, 18.10, 18.11, and 18.12), 21 (except 21.1.2 and 21.2.3), 22, 24, 25, 26, of the technical report titled "Stibnite Gold Project, Prefeasibility Study Technical Report, Valley County, Idaho" (the "Technical Report"), with an effective date of December 8, 2014 and amended March 28, 2019, prepared for Midas Gold Corporation.
8. I visited the project site on July 18, 2017. I have had no prior involvement with the property that is subject of the Technical Report.
9. As of the effective date of the technical report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 28th day of March, 2019.

(Signed and Sealed) "*Lee A. Becker*"
Signature of Qualified Person

Lee A. Becker, P.E., P.Eng.
Print Name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Richard K Zimmerman, R.G., do hereby certify that:

1. I am currently employed as a Registered Professional Geologist by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, Ste. 101
   Tucson, Arizona 85704
   U.S.A.

2. I am a graduate of Carleton College and received a Bachelor of Arts degree in Geology in 1976. I am also a graduate of the University of Michigan and received a M.Sc. degree in Geology 1980.

3. I am a:

   - Registered Professional Geology in good standing with the State of Arizona (No. 24064)
   - Registered Member in good standing of the Society for Mining, Metallurgy and Exploration, Inc. (No. 3612900RM)

4. I have practiced geology, mineral exploration, environmental remediation, and project management for 39 years. I have worked for mining and exploration companies for 9 years and engineering consulting firms for 22 years. The past 8 years have been spent with M3 Engineering & Technology Corporation managing, planning, and constructing processing plants for base and precious metals.

5. I have read National Instrument 43-101 (NI 43-101) and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

6. I have read the definition of “qualified person” set out NI 43-101 and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.

7. I am responsible for Sections 1, 2, 3, 4, 5, 6, 19, 23, 24, 25, 26, and 27 of the technical report titled “Stibnite Gold Project, Prefeasibility Study Technical Report, Valley County, Idaho” (the “Technical Report”), with an effective date of December 8, 2014 and amended March 28, 2019, prepared for Midas Gold Corporation.

8. I visited the project site on March 7, 2013. I have had no prior involvement with the property that is subject of the Technical Report.

9. As of the effective date of the technical report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all tests in section 1.5 of National Instrument 43-101.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them of the Technical Report for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public.

Dated this 28 day of March, 2019

(Signed) (Sealed) *“Richard K Zimmerman”*
Signature of Qualified Person

Richard K Zimmerman, M.Sc., R.G., SME-RM No. 3612900RM
Print Name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Garth Kirkham, P.Geo, do hereby certify that:

1. I am currently employed as a Consulting Geoscientist and Principal for:

   Kirkham Geosystems Ltd.
   6331 Palace Place
   Burnaby, BC, Canada V5E 1Z6

2. I am a graduate of the University of Alberta in 1983 with a BSc. I have continuously practiced my profession since 1988. I have worked on and been involved with NI43-101 studies on the Kutcho, Adi Nefas, Debarwa, Tahuehueto, Demir deposits.

3. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC).

4. I have read the definition of "qualified person" set out in National instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I am responsible for Sections 7, 8, 9, 10, 11, 12, and 14 of the technical report titled "Stibnite Gold Project, Prefeasibility Study Technical Report, Valley County, Idaho", dated effective December 8, 2014 and amended March 28, 2019.

6. I have not had prior involvement with the property that is the subject of the Technical Report.

7. I have visited the Stibnite Gold property on April 23-25, 2014 and July 14-15, 2014.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 28th day of March, 2019.

(Signed) (Sealed) *"Garth Kirkham"*
Signature of Qualified Person

Garth Kirkham, P.Geo.
Print Name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Christopher Martin, MIMMM, C.Eng. do hereby certify that:

1. I am currently employed as Principal Metallurgist by Blue Coast Metallurgy, Ltd, 1020 Herring Gull Way, Parksville, BC V9P 1R2

2. I hold degrees in Mineral Processing Technology from Camborne School of Mines (BSc(Hons)) (1984) and Metallurgical Engineering from McGill University (1988).

3. I am a full professional member of the Institute of Minerals, Materials, and Mining, in good standing since 1990.

4. I have practiced my profession in plant operations, in flowsheet development, plant design and optimization since 1984.

5. I have read the definition of "qualified person" set out in National instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for Section 13 of the technical report titled "Stibnite Gold Project, Prefeasibility Study Technical Report, Valley County, Idaho", dated effective December 8, 2014 and amended March 28, 2019.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I have visited the Stibnite Gold property on August 25, 2011 for one day.

9. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 28th day of March, 2019.

(Signed) (Sealed) *"Christopher Martin"*
Signature of Qualified Person

Christopher Martin, MIMMM, C.Eng.
Print Name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, John M. Marek P.E. do hereby certify that:

a) I am currently employed as the President and a Senior Mining Engineer by:

Independent Mining Consultants, Inc.
3560 E. Gas Road
Tucson, Arizona, USA 85714

b) This certificate is part of the report titled "Stibnite Gold Project, Prefeasibility Study Technical Report, Valley County, Idaho", with an effective date of 8 December 2014 and amended March 28, 2019.

c) I graduated with the following degrees from the Colorado School of Mines
    Bachelors of Science, Mineral Engineering – Physics 1974
    Masters of Science, Mining Engineering 1976

d) I am a Registered Professional Mining Engineer in the State of Arizona USA
   Registration # 12772

- I am a Registered Professional Engineer in the State of Colorado USA
  Registration # 16191
- I am a Registered Member of the American Institute of Mining and Metallurgical Engineers, Society of Mining Engineers

e) I have worked as a mining engineer, geoscientist, and reserve estimation specialist for more than 37 years. I have managed drill programs, overseen sampling programs, and interpreted geologic occurrences in both precious metals and base metals for numerous projects over that time frame. My advanced training at the university included geostatistics and I have built upon that initial training as a resource modeler and reserve estimation specialist in base and precious metals for my entire career. I have acted as the Qualified Person on these topics for numerous Technical Reports.

f) My work experience includes mine planning, equipment selection, mine cost estimation and mine feasibility studies for base and precious metals projects worldwide for over 37 years. I have experience with the overall management, review, and assembly of feasibility studies.

g) I last visited the Stibnite Gold project site on 17 September 2013. One day was spent reviewing geology and operating conditions at site.

h) I am responsible for the following sections of the report titled "Stibnite Gold Project, Prefeasibility Technical Report, Valley County, Idaho", with an effective date of 8 December 2014: 15, 16, 21.1.2, and 21.2.3. I contributed to the Section 1 Summary.

i) I am independent of Midas Gold Corp. applying the tests in Section 1.5 of National Instrument 43-101.

j) Independent Mining Consultants, Inc. and John Marek have not worked on the Stibnite Gold Project previous to this report.

k) I have read National Instrument 43-101 and Form 43-101F1, and to my knowledge, the Technical Report has been prepared in compliance with that instrument and form.

l) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: March 28, 2019.

(Signed) (Sealed) *"John M. Marek"*
Signature of Qualified Person, Registered Member of the American Institute of Mining and Metallurgical Engineers, Society of Mining Engineers

John M. Marek, P.E.
Print Name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Allen R. Anderson P.E., do hereby certify that:

1. I am currently employed as President of Allen R. Anderson Metallurgical Engineer Inc. located at:
11050 E. Ft. Lowell Rd.
Tucson AZ, 85749
2. I am a graduate of South Dakota School of Mines and Technology, May 1977, and hold Bachelor of Science degree in Metallurgical Engineering.
3. I am a Registered Professional Engineer – Mining in the State of Arizona in good standing. Registration # 50635.
4. I have been employed in the mining industry for thirty-seven years. I worked for mining and exploration companies including DUVAL and Battle Mountain Gold for twenty years and for engineering companies including The Winters Company, Jacobs Engineering and KD Engineering for seven years. I have been working as an independent consultant for the last ten years.
5. I have read the definition of "qualified person" set out in National instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for Section 17 of the technical report titled "Stibnite Gold Project, Prefeasibility Study Technical Report, Valley County, Idaho", dated effective December 8, 2014 (the "Technical Report") and amended March 28, 2019.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I have not visited the Stibnite Gold property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.
10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101; and Form 43-101F1 and the Technical Report. Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 28th day of March, 2019.

(Signed) (Sealed) *"Allen R. Anderson"*
Signature of Qualified Person

Allen R. Anderson, P.E.
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Richard C. Kinder, P.E., do hereby certify that:

1. I am currently employed as an Engineer and Project Manager by:

   HDR Engineering, Inc.
   412 E. Parkcenter Blvd., Suite 100
   Boise, Idaho 83706-6659

2. I was awarded the degree of Bachelor of Science in Civil Engineering from Washington State University in 1984.

3. I am a Registered Professional Engineer in good standing in the State of Idaho (#7277)

4. I have practiced engineering and project management for the past 29 years in civil engineering and construction administration of transportation projects across the United States for several construction and engineering firms.

5. I have read the definition of "qualified person" set out in National instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for Section 18.2 of the technical report titled "Stibnite Gold Project, Prefeasibility Study Technical Report, Valley County, Idaho", dated effective December 8, 2014 (the "Technical Report") and amended March 28, 2019.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I visited the Stibnite Gold property and surrounding area on several occasions from September 2011 through October 2012 for the purpose of conducting an analysis for alternative access to the property to support the Technical Report. The last site trip was conducted on October 12, 2012 for duration of approximately 4 hours to explore the Burntlog Alternative access route. I also coordinated with Midas Gold thereafter on issues related to the alternative access roads.

9. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101, Form 43-101F1, and the Technical Report. The Section 18.2 has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 28th day of March, 2019.

(Signed) (Sealed) *"Richard C. Kinder"*
Signature of Qualified Person

Richard C. Kinder, P.E.
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Peter E. Kowalewski P.E., do hereby certify that:

1. I am currently employed as a Principal Engineer by Tierra Group International, Ltd. ("Tierra Group") with an office at 111 East Broadway, Suite 220, Salt Lake City, Utah, U.S.A.

2. I am a graduate of the Colorado School of Mines in 1992 and 1997 with B.Sc. (Geological Engineering) and M.E. (Applied Mechanics) degrees, respectively. I have practiced my profession continuously since graduation in 1992, focusing on the civil, geotechnical, hydrologic, and hydraulic design of facilities primarily for the mining industry. My primary focus has been on the design, permitting, construction, operation, and closure of mine waste containment facilities such as tailings impoundments, heap leach facilities, waste rock storage facilities, and appurtenant structures such as ponds and channels.

3. I am a Licensed Professional Engineer (P.E.) (Civil) in multiple States, including the State of Idaho (Idaho License #15289). In addition, I am a Registered Member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME) (Member #4055322RM).

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

5. I am responsible for Sections 18.9, 18.10, 18.11, 18.12, 20 of the technical report titled "Stibnite Gold Project, Prefeasibility Study Technical Report, Valley County, Idaho", dated effective December 8, 2014 and amended March 28, 2019.

6. I have not had prior involvement with the property that is the subject of the Technical Report.

7. I visited the Stibnite Gold property on March 7, 2013.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

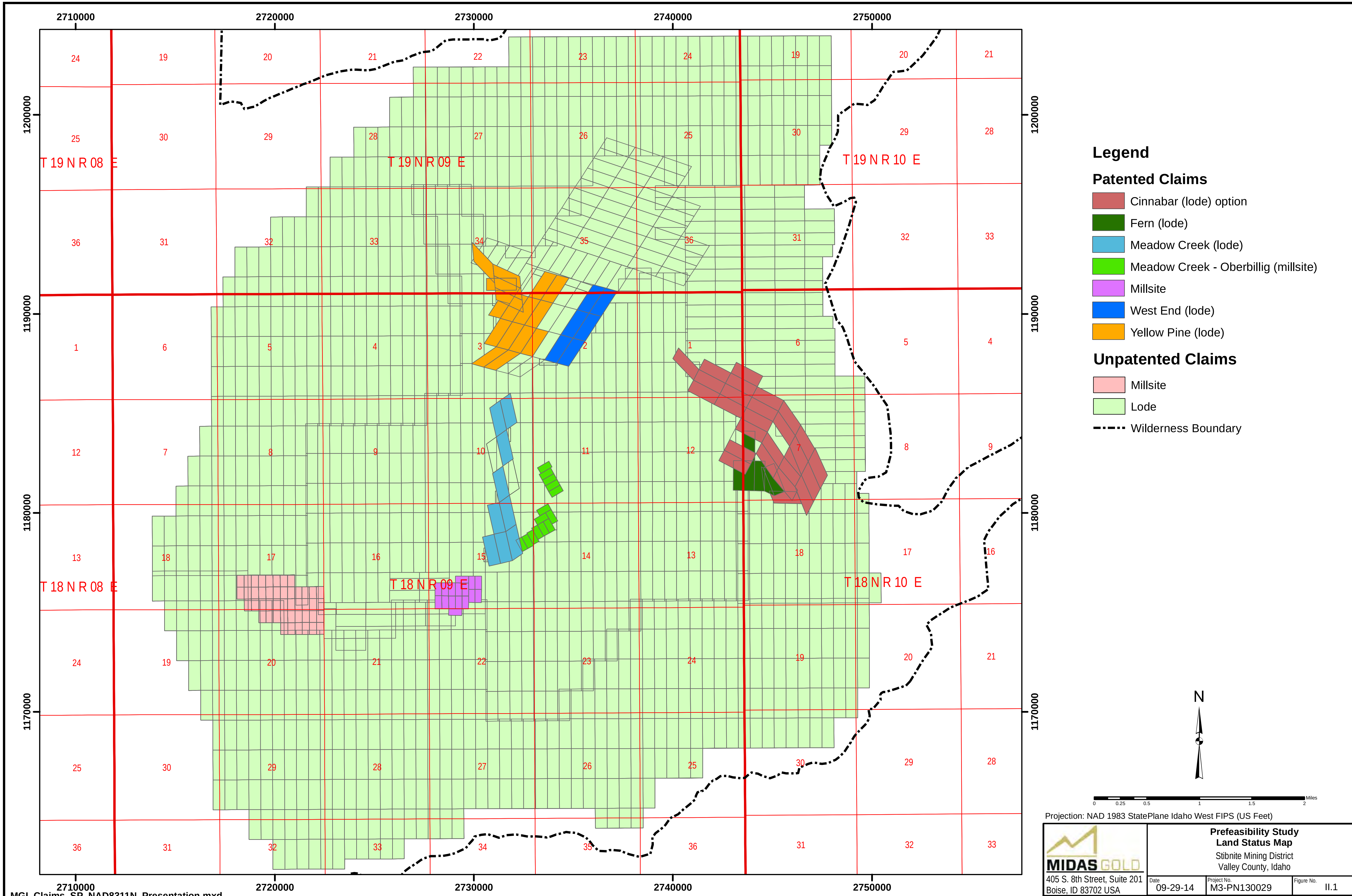
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 28th day of March, 2019.

(Signed) (Sealed) *"Peter E. Kowalewski"*
Signature of Qualified Person

Peter E. Kowalewski, P.E.
Print name of Qualified Person

APPENDIX II: PROPERTY DESCRIPTION AND LOCATION



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2710000
2720000
2730000
2740000
2750000
1200000
1190000
1180000
1170000
Legend
Patented Claims
Cinnabar (lode) option
Fern (lode)
Meadow Creek (lode)
Meadow Creek - Oberbillig (millsite)
Millsite
West End (lode)
Yellow Pine (lode)
Unpatented Claims
Millsite
Lode
Wilderness Boundary
N
Miles
Projection: NAD 1983 StatePlane Idaho West FIPS (US Feet)
MIDAS GOLD
405 S. 8th Street, Suite 201
Boise, ID 83702 USA
Prefeasibility Study
Land Status Map
Stibnite Mining District
Valley County, Idaho
Date
09-29-14
Project No.
M3-PN130029
Figure No.
II.1
MGI_Claims_SP_NAD8311N_Presentation.mxd

Table II.1: Mineral Concession Summary[8]

PATENTED CLAIMS						
Acquisition	Claim Group	Number of Claims		Acres	Hectares	Property Tax 2013
		Lode	Millsite			
2009 Bradley[1,7]	Meadow Creek	9	--	184.4	74.6	$135.70
2009 JJO[2]	Meadow Creek – Oberbillig[6]	--	14	68.5	27.7	$825.06
	Millsite	--	16	80.5	32.6	$58.86
	West End	6	--	123.9	50.2	$95.60
2011 Yellow Pine[3]	Yellow Pine	17	--	300.7	121.7	$220.78
2011 Fern[4]	Fern	6	--	99.7	40.4	$73.50
2011 Cinnabar[5]	Cinnabar	27	--	484.8	196.2	$355.96
	Totals	65	30	1,342.6	543.3	$1,765.46
UNPATENTED CLAIMS						
Acquisition	Claim Group	Number of Claims		Acres	Hectares	BLM Claims Fees
		Lode	Millsite			
2009 acquisition	Unpatented	183	46			$32,060
2009 staked	Unpatented	238	--			$33,320
2011 Yellow Pine[3]	Unpatented	8	--			$1,120
2011 staked	Unpatented	921	--			$128,940
2012 staked	Unpatented	1	--			$140
	Totals	1,351	46	25,761.5	10,425.3	$195,580

Notes:
1. *Acquired by 3/31/2008 Option to Purchase from Bradley Mining Co., warranty deed 6/11/2009.*
2. *Acquired from JJO, LLC via 6/2/2009 Promissory Note, matures 6/2/2015 (balance remaining $40,000).*
3. *Acquired from Bradley Mining Company via 11/7/2003 Option to Purchase, warranty deed 11/28/2012.*
4. *Acquired from Stephens, warranty deed 4/28/2011.*
5. *Acquired from JJO, LLC via 5/1/2011 Option to Purchase, five annual extensions expire 5/1/17 (balance remaining $300,000).*
6. *Hecla Mining Co. retains surface estate on portion of claims MS #1 through MS #6.*
7. *5% NSR royalty right acquired via Promissory Note, matures 6/2/15 (balance remaining $160,000).*
8. *The entire Golden Meadows property (excluding the Cinnabar group) is subject to the May 9, 2013 1.7% NSR precious metals royalty held Franco-Nevada.*

Table II.2: Mineral Concession Summary – Patented Claims

Count	Claim Name	Claim Type	Mine Survey No.	Date Patented	Patent No.	Section, Township, Range	PIN[1]	Owner[2]	Date Acquired
Meadow Creek - Oberbillig[4]									
1	MS 1	Millsite	3655	09/27/1990	11900097	15-18N-9E	RP18N09E115495	MGI[3]	04/28/2009
2	MS 2	Millsite	3655	09/27/1990	11900097	14,15-18N-9E	RP18N09E115495	MGI[3]	04/29/2009
3	MS 3	Millsite	3655	09/27/1990	11900097	14,15-18N-9E	RP18N09E115495	MGI[3]	04/30/2009
4	MS 4	Millsite	3655	09/27/1990	11900097	14,15-18N-9E	RP18N09E115495	MGI[3]	05/01/2009
5	MS 5	Millsite	3655	09/27/1990	11900097	14-18N-9E	RP18N09E115495	MGI[3]	05/02/2009
6	MS 6	Millsite	3655	09/27/1990	11900097	14-18N-9E	RP18N09E115495	MGI[3]	05/03/2009
7	MS 7	Millsite	3655	09/27/1990	11900097	14-18N-9E	RP18N09E115495	MGI	05/04/2009
8	MS 8	Millsite	3655	09/27/1990	11900097	14-18N-9E	RP18N09E115495	MGI	05/05/2009
9	MS 9	Millsite	3655	09/27/1990	11900097	14-18N-9E	RP18N09E115495	MGI	05/06/2009
10	MS 13	Millsite	3655	09/27/1990	11900097	11-18N-9E	RP18N09E115495	MGI	05/07/2009
11	MS 14	Millsite	3655	09/27/1990	11900097	11-18N-9E	RP18N09E115495	MGI	05/08/2009
12	MS 15	Millsite	3655	09/27/1990	11900097	11-18N-9E	RP18N09E115495	MGI	05/09/2009
13	MS 16	Millsite	3655	09/27/1990	11900097	11-18N-9E	RP18N09E115495	MGI	05/10/2009
14	MS 17	Millsite	3655	09/27/1990	11900097	11-18N-9E	RP18N09E115495	MGI	05/11/2009
Meadow Creek[5]									
15	Meadow Creek No. 3	Lode	3039	02/19/1926	974550	15-18N-9E	RP18N09E108995	MGIAC	05/12/2009
16	Meadow Creek No. 4	Lode	3039	02/19/1926	974550	15-18N-9E	RP18N09E108995	MGIAC	05/13/2009
17	Meadow Creek No. 2	Lode	3039	02/19/1926	974550	15-18N-9E	RP18N09E108995	MGIAC	05/14/2009
18	Meadow Creek No. 1	Lode	3039	02/19/1926	974550	15-18N-9E	RP18N09E108995	MGIAC	05/15/2009
19	Meadow Creek No. 5	Lode	3039	02/19/1926	974550	15-18N-9E	RP18N09E108995	MGIAC	05/16/2009
20	Monday No. 6	Lode	3397	01/25/1946	1120542	10-18N-9E	RP18N09E038995	MGIAC	05/17/2009
21	Meadow Creek No. 8	Lode	3397	01/25/1946	1120542	10,15-18N-9E	RP18N09E038995	MGIAC	05/18/2009
22	Monday No. 2	Lode	3397	01/25/1946	1120542	10-18N-9E	RP18N09E038995	MGIAC	05/19/2009
23	Monday No. 3	Lode	3397	01/25/1946	1120542	10-18N-9E	RP18N09E038995	MGIAC	05/20/2009
West End[4]									
24	MW 9	Lode	3645	12/15/1989	11900012	2-18N-9E	RP18N09E020026	MGI	05/21/2009

Count	Claim Name	Claim Type	Mine Survey No.	Date Patented	Patent No.	Section, Township, Range	PIN[1]	Owner[2]	Date Acquired
25	MW 13	Lode	3645	02/25/1987	11870032	2-18N-9E	RP18N09E020026	MGI	05/22/2009
26	MW 22	Lode	3645	02/25/1987	11870032	2-18N-9E	RP18N09E020026	MGI	05/23/2009
27	MW 8	Lode	3645	12/15/1989	11900012	2-18N-9E	RP18N09E020026	MGI	05/24/2009
28	MW 12	Lode	3645	02/25/1987	11870032	2-18N-9E; 35-19N-9E	RP18N09E020026	MGI	05/25/2009
29	MW 23	Lode	3645	02/25/1987	11870032	2-18N-9E; 35-19N-9E	RP18N09E020026	MGI	05/26/2009
Millsite[4]									
30	MS 36	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	05/27/2009
31	MS 51	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	05/28/2009
32	MS 52	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	05/29/2009
33	MS 53	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	05/30/2009
34	MS 28	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	05/31/2009
35	MS 35	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	06/01/2009
36	MS 37	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	06/02/2009
37	MS 38	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	06/03/2009
38	MS 39	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	06/04/2009
39	MS 42	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	06/05/2009
40	MS 43	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	06/06/2009
41	MS 44	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	06/07/2009
42	MS 45	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	06/08/2009
43	MS 46	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	06/09/2009
44	MS 47	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	06/10/2009
45	MS 48	Millsite	No Survey	09/27/1990	11900098	15,22-18N-9E	RP18N09E155300	MGI	06/11/2009
Fern[6]									
47	Spruce Grove	Lode	3030	11/04/1926	988370	12-18N-9E	RP18N09E127345	MGIAC	04/28/2011
48	Fern	Lode	3030	11/04/1926	988370	7-18N-10E	RP18N09E127345	MGIAC	04/28/2011
49	Fern No. 2	Lode	3030	11/04/1926	988370	7-18N-10E	RP18N09E127345	MGIAC	04/28/2011
50	Fern No. 4	Lode	3030	11/04/1926	988370	7-18N-10E	RP18N09E127345	MGIAC	04/28/2011
51	Bucks Bed No. 1	Lode	3030	11/04/1926	988370	7-18N-10E	RP18N09E127345	MGIAC	04/28/2011

Count	Claim Name	Claim Type	Mine Survey No.	Date Patented	Patent No.	Section, Township, Range	PIN[1]	Owner[2]	Date Acquired
52	Bucks Bed	Lode	3030	11/04/1926	988370	7-18N-10E	RP18N09E127345	MGIAC	04/28/2011
Cinnabar[7]									
53	Hermes	Lode	3038	06/02/1927	1003392	1,12-18N-9E; 6,7-18N-10E	RP18N09E018435	JJO	option
54	Annie Sell	Lode	3038	06/02/1927	1003392	7-18N-10E	RP18N09E018435	JJO	option
55	Pretty Maid	Lode	3038	06/02/1927	1003392	1-18N-9E	RP18N09E018435	JJO	option
56	West End	Lode	3395	07/15/1946	1121067	1-18N-9E	RP18N09E013840	JJO	option
57	Liberty	Lode	3395	07/15/1946	1121067	1,12-18N-9E	RP18N09E013840	JJO	option
58	Liberty No. 1	Lode	3395	07/15/1946	1121067	1,12-18N-9E; 7-18N-10E	RP18N09E013840	JJO	option
59	U.S.A.	Lode	3395	07/15/1946	1121067	12-18N-9E; 7-18N-10E	RP18N09E013840	JJO	option
60	Vermillion Ext. No. 2	Lode	3395	07/15/1946	1121067	7-18N-10E; 12-18N-9E	RP18N09E013840	JJO	option
61	Golden Gate No. 4	Lode	3038	06/02/1927	1003392	7-18N-10E	RP18N09E018435	JJO	option
62	Vermillion Ext. No. 1	Lode	3038	06/02/1927	1003392	7-18N-10E	RP18N10E071525	JJO	option
63	Vermillion	Lode	3038	06/02/1927	1003392	7,18-18N-10E	RP18N10E071525	JJO	option
64	Monumental No. 1	Lode	3396	11/01/1944	1119154	1-18N-9E	RP18N09E018150	JJO	option
65	Monumental No. 2	Lode	3396	11/01/1944	1119154	1-18N-9E; 6,7-18N-10E	RP18N09E018150	JJO	option
66	C 12	Lode	3396	11/01/1944	1119154	6-18N-10E; 1-18N-9E	RP18N09E018150	JJO	option
67	Monumental No. 3	Lode	3396	11/01/1944	1119154	6,7-18N-10E	RP18N09E018150	JJO	option
68	Monumental No. 4	Lode	3396	11/01/1944	1119154	7-18N-10E	RP18N09E018150	JJO	option
69	Monumental No. 5	Lode	3396	11/01/1944	1119154	7-18N-10E	RP18N09E018150	JJO	option
70	Monumental No. 6	Lode	3396	11/01/1944	1119154	7-18N-10E	RP18N09E018150	JJO	option
71	White Metal No. 1	Lode	3395	07/15/1946	1121067	7-18N-10E	RP18N09E122155	JJO	option
72	Vermillion Ext. No. 3	Lode	3395	07/15/1946	1121067	7-18N-10E	RP18N09E013840	JJO	option
73	White Metal No. 2	Lode	3395	07/15/1946	1121067	7-18N-10E	RP18N09E122155	JJO	option
74	Flyer	Lode	3395	07/15/1946	1121067	7-18N-10E; 18-18N-10E	RP18N09E122155	JJO	option
75	White Metal	Lode	3395	07/15/1946	1121067	7-18N-10E	RP18N09E013840	JJO	option
76	White Metal No. 3	Lode	3395	07/15/1946	1121067	7-18N-10E	RP18N09E122155	JJO	option
46	White Metal No. 4	Lode	3395	07/15/1946	1121067	7,18-18N-10E	RP18N09E122155	JJO	option
77	White Metal No. 6	Lode	3395	07/15/1946	1121067	12-18N-9E; 7-18N-10E	RP18N09E122155	JJO	option

Count	Claim Name	Claim Type	Mine Survey No.	Date Patented	Patent No.	Section, Township, Range	PIN[1]	Owner[2]	Date Acquired
78	Mountain Belle Frac.	Lode	3395	07/15/1946	1121067	7-18N-10E; 12-18N-9E	RP18N09E013840	JJO	option
Yellow Pine[8]									
79	Hennessy No. 6	Lode	3357	06/18/1941	1111588	2,3-18N-9E	RP18N09E030005	IGR	04/28/2009
80	Homestake No. 5	Lode	3357	06/18/1941	1111588	2,3-18N-9E	RP18N09E030005	IGR	04/28/2009
81	Hennessy No. 2	Lode	3357	06/18/1941	1111588	3-18N-9E	RP18N09E030005	IGR	04/28/2009
82	Hennessy Lode No. 4	Lode	3357	06/18/1941	1111588	3-18N-9E	RP18N09E030005	IGR	04/28/2009
83	Hennessy No. 3	Lode	3357	06/18/1941	1111588	3-18N-9E	RP18N09E030005	IGR	04/28/2009
84	Hennessy No. 5	Lode	3357	06/18/1941	1111588	3-18N-9E	RP18N09E030005	IGR	04/28/2009
85	Homestake No. 1	Lode	3357	06/18/1941	1111588	2,3-18N-9E	RP18N09E030005	IGR	04/28/2009
86	Hennessy No. 1	Lode	3357	06/18/1941	1111588	2,3-18N-9E	RP18N09E030005	IGR	04/28/2009
87	Hennessy Lode No. 7	Lode	3357	06/18/1941	1111588	3-18N-9E	RP18N09E030005	IGR	04/28/2009
88	Homestake No. 2	Lode	3357	06/18/1941	1111588	2,3-18N-9E; 35-19N-9E	RP18N09E030005	IGR	04/28/2009
89	Homestake	Lode	3357	06/18/1941	1111588	2,3-18N-9E; 35-19N-9E	RP18N09E030005	IGR	04/28/2009
90	A No. 1	Lode	3246	05/12/1933	1064103	3-18N-9E	RP18N09E030020	IGR	04/28/2009
91	Fair Deal No. 3	Lode	3246	05/12/1933	1064103	3-18N-9E; 34-19N-9E	RP18N09E030020	IGR	04/28/2009
92	Fair Deal No. 2	Lode	3246	05/12/1933	1064103	34-19N-9E	RP18N09E030020	IGR	04/28/2009
93	Fair Deal No. 1	Lode	3246	05/12/1933	1064103	34-19N-9E	RP18N09E030020	IGR	04/28/2009
94	Fair Deal No. 4	Lode	3246	05/12/1933	1064103	34-19N-9E	RP18N09E030020	IGR	04/28/2009
95	Camp Bird No. 2	Lode	3246	05/12/1933	1064103	34-19N-9E	RP18N09E030020	IGR	04/28/2009

Notes:
1. *PIN = Valley County Parcel Identification Number.*
2. *MGI = Midas Gold, Inc., MGIAC = MGI Acquisition Corp., IGR = Idaho Gold Resources LLC, JJO = J.J. Oberbillig Estate.*
3. *Split estate - MGI owns mineral rights, Hecla Mining Co. owns surface on a portion of these six claims.*
4. *Part of Oberbillig claim group included in MGI purchase from JJO, LLC, closed escrow June 2, 2009.*
5. *Meadow Creek claim group acquired from Bradley Mining Co. via Mar. 31, 2008 Option to Purchase, warranty deed June 11, 2009.*
6. *Fern claim group acquired from Stevens, warranty deed April 28, 2011.*
7. *Cinnabar group included in MGI Acquisition Corporation Option to Purchase from JJO, LLC dated May 1, 2011, five annual extensions that expire May 1, 2017.*
8. *Yellow Pine claim group acquired from Bradley Mining Co. via Nov. 7, 2003 Option to Purchase, closed escrow Nov. 29, 2012.*

Table II.3: Mineral Concession Summary – Unpatented Claims

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1	SFMS 1	190070	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312095	1
2	SFMS 2	190071	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312096	2
3	SFMS 3	190072	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312097	3
4	SFMS 4	190073	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312098	4
5	SFMS 5	190074	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312099	5
6	SFMS 6	190075	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312100	6
7	SFMS 7	190076	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312101	7
8	SFMS 8	190077	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312102	8
9	SFMS 9	190078	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312103	9
10	SFMS 10	190079	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312104	10
11	SFMS 11	190080	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312105	11
12	SFMS 12	190081	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312106	12
13	SFMS 13	190082	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312107	13
14	SFMS 14	190083	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312108	14
15	SFMS 15	190084	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312109	15
16	SFMS 16	190085	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312110	16
17	SFMS 17	190086	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312111	17
18	SFMS 18	190087	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312112	18
19	SFMS 19	190088	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312113	19
20	SFMS 20	190089	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312114	20
21	SFMS 21	190090	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312115	21
22	SFMS 22	190091	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312116	22
23	SFMS 23	190092	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312117	23
24	SFMS 24	190093	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312118	24
25	SFMS 25	190094	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312119	25
26	SFMS 26	190095	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312120	26
27	SFMS 27	190096	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312121	27

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
28	SFMS 28	190097	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312122	28
29	SFMS 29	190098	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312123	29
30	SFMS 30	190099	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312124	30
31	SFMS 31	190100	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312125	31
32	SFMS 32	190101	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312126	32
33	SFMS 33	190102	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312127	33
34	SFMS 34	190103	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312128	34
35	SFMS 35	190104	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312129	35
36	SFMS 36	190105	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312130	36
37	SFMS 37	190106	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312131	37
38	SFMS 38	190107	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312132	38
39	SFMS 39	190108	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312133	39
40	SFMS 40	190109	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312134	40
41	SFMS 41	190110	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312135	41
42	SFMS 42	190111	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312154	42
43	SFMS 43	190112	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312155	43
44	SFMS 44	190113	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312156	44
45	SFMS 45	190114	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312157	45
46	SFMS 46	190115	Millsite	MGI	07/27/2006	Niagara	08/14/2006	312158	46
47	SF 1	189924	Lode	MGI	05/24/2006	Niagara	08/10/2006	311895	47
48	SF 2	189925	Lode	MGI	05/24/2006	Niagara	08/10/2006	311896	48
49	SF 3	189926	Lode	MGI	05/24/2006	Niagara	08/10/2006	311897	49
50	SF 4	189927	Lode	MGI	05/24/2006	Niagara	08/10/2006	311898	50
51	SF 5	189928	Lode	MGI	05/24/2006	Niagara	08/10/2006	311899	51
52	SF 6	189929	Lode	MGI	05/24/2006	Niagara	08/10/2006	311990	52
53	SF 7	189930	Lode	MGI	05/24/2006	Niagara	08/10/2006	311901	53
54	SF 8	189931	Lode	MGI	05/24/2006	Niagara	08/10/2006	311902	54
55	SF 9	189932	Lode	MGI	05/24/2006	Niagara	08/10/2006	311903	55

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
56	SF 10	189933	Lode	MGI	05/24/2006	Niagara	08/10/2006	311904	56
57	SF 11	189934	Lode	MGI	05/24/2006	Niagara	08/10/2006	311905	57
58	SF 12	189935	Lode	MGI	05/24/2006	Niagara	08/10/2006	311906	58
59	SF 13	189936	Lode	MGI	05/24/2006	Niagara	08/10/2006	311907	59
60	SF 14	189937	Lode	MGI	05/24/2006	Niagara	08/10/2006	311911	60
61	SF 15	189938	Lode	MGI	05/24/2006	Niagara	08/10/2006	311912	61
62	SF 16	189939	Lode	MGI	05/24/2006	Niagara	08/10/2006	311913	62
63	SF 17	189940	Lode	MGI	05/24/2006	Niagara	08/10/2006	311914	63
64	SF 18	189941	Lode	MGI	05/24/2006	Niagara	08/10/2006	311915	64
65	SF 19	189942	Lode	MGI	05/24/2006	Niagara	08/10/2006	311916	65
66	SF 20	189943	Lode	MGI	05/24/2006	Niagara	08/10/2006	311917	66
67	SF 21	189944	Lode	MGI	05/24/2006	Niagara	08/10/2006	311918	67
68	SF 22	189945	Lode	MGI	05/24/2006	Niagara	08/10/2006	311919	68
69	SF 23	189946	Lode	MGI	05/24/2006	Niagara	08/10/2006	311920	69
70	SF 24	189947	Lode	MGI	05/24/2006	Niagara	08/10/2006	311921	70
71	SF 25	189948	Lode	MGI	05/24/2006	Niagara	08/10/2006	311922	71
72	SF 26	189949	Lode	MGI	05/24/2006	Niagara	08/10/2006	311923	72
73	SF 27	189950	Lode	MGI	05/24/2006	Niagara	08/10/2006	311924	73
74	SF 28	189951	Lode	MGI	05/24/2006	Niagara	08/10/2006	311925	74
75	SF 29	189952	Lode	MGI	05/24/2006	Niagara	08/10/2006	311926	75
76	SF 30	189953	Lode	MGI	05/24/2006	Niagara	08/10/2006	311927	76
77	SF 31	189954	Lode	MGI	05/24/2006	Niagara	08/11/2006	311949	77
78	SF 32	189955	Lode	MGI	05/24/2006	Niagara	08/11/2006	311950	78
79	SF 33	189956	Lode	MGI	05/24/2006	Niagara	08/11/2006	311951	79
80	SF 34	189957	Lode	MGI	05/24/2006	Niagara	08/11/2006	311952	80
81	SF 35	189958	Lode	MGI	05/24/2006	Niagara	08/11/2006	311953	81
82	SF 36	189959	Lode	MGI	05/24/2006	Niagara	08/11/2006	311954	82
83	SF 37	189960	Lode	MGI	05/24/2006	Niagara	08/11/2006	311955	83

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
84	SF 38	189961	Lode	MGI	05/24/2006	Niagara	08/11/2006	311956	84
85	SF 39	189962	Lode	MGI	05/24/2006	Niagara	08/11/2006	311957	85
86	SF 40	189963	Lode	MGI	05/24/2006	Niagara	08/11/2006	311958	86
87	SF 41	189964	Lode	MGI	05/24/2006	Niagara	08/11/2006	311959	87
88	SF 42	189965	Lode	MGI	05/24/2006	Niagara	08/11/2006	311960	88
89	SF 43	189966	Lode	MGI	05/24/2006	Niagara	08/11/2006	311961	89
90	SF 44	189967	Lode	MGI	05/24/2006	Niagara	08/11/2006	311962	90
91	SF 45	189968	Lode	MGI	05/24/2006	Niagara	08/11/2006	311963	91
92	SF 46	189969	Lode	MGI	05/24/2006	Niagara	08/11/2006	311964	92
93	SF 47	189970	Lode	MGI	05/24/2006	Niagara	08/11/2006	311965	93
94	SF 48	189971	Lode	MGI	05/24/2006	Niagara	08/11/2006	311966	94
95	SF 49	189972	Lode	MGI	05/24/2006	Niagara	08/11/2006	311967	95
96	SF 50	189973	Lode	MGI	05/24/2006	Niagara	08/11/2006	311968	96
97	SF 52	189974	Lode	MGI	05/24/2006	Niagara	08/11/2006	311969	97
98	SF 53	189975	Lode	MGI	05/24/2006	Niagara	08/11/2006	311970	98
99	SF 54	189976	Lode	MGI	05/24/2006	Niagara	08/11/2006	311971	99
100	SF 55	189977	Lode	MGI	05/24/2006	Niagara	08/11/2006	311972	100
101	SF 56	189978	Lode	MGI	05/24/2006	Niagara	08/11/2006	311973	101
102	SF 57	189979	Lode	MGI	05/24/2006	Niagara	08/11/2006	311974	102
103	SF 58	189980	Lode	MGI	05/24/2006	Niagara	08/11/2006	311975	103
104	SF 59	189981	Lode	MGI	05/24/2006	Niagara	08/11/2006	311976	104
105	SF 61	189982	Lode	MGI	05/24/2006	Niagara	08/11/2006	311977	105
106	SF 62	189983	Lode	MGI	05/24/2006	Niagara	08/11/2006	311978	106
107	SF 63	199733	Lode	MGI	09/16/2009	MGIAC	11/13/2009	347151	107
108	SF 64	199734	Lode	MGI	09/16/2009	MGIAC	11/13/2009	347152	108
109	SF 65	189986	Lode	MGI	05/24/2006	Niagara	08/11/2006	311981	109
110	SF 66	189987	Lode	MGI	05/24/2006	Niagara	08/11/2006	311982	110
111	SF 67	189988	Lode	MGI	05/24/2006	Niagara	08/11/2006	311983	111

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
112	SF 68	189989	Lode	MGI	05/24/2006	Niagara	08/11/2006	312022	112
113	SF 69	189990	Lode	MGI	05/24/2006	Niagara	08/11/2006	312023	113
114	SF 70	189991	Lode	MGI	05/24/2006	Niagara	08/11/2006	312024	114
115	SF 71	199735	Lode	MGI	09/16/2009	MGIAC	11/13/2009	347153	115
116	SF 72	199736	Lode	MGI	09/16/2009	MGIAC	11/13/2009	347154	116
117	SF 73	189994	Lode	MGI	05/24/2006	Niagara	08/11/2006	312027	117
118	SF 74	189995	Lode	MGI	05/24/2006	Niagara	08/11/2006	312028	118
119	SF 75	189996	Lode	MGI	05/24/2006	Niagara	08/11/2006	312029	119
120	SF 76	189997	Lode	MGI	05/24/2006	Niagara	08/11/2006	312030	120
121	SF 77	189998	Lode	MGI	05/24/2006	Niagara	08/11/2006	312031	121
122	SF 78	189999	Lode	MGI	05/24/2006	Niagara	08/11/2006	312032	122
123	SF 79	190000	Lode	MGI	05/24/2006	Niagara	08/11/2006	312033	123
124	SF 80	190001	Lode	MGI	05/24/2006	Niagara	08/11/2006	312034	124
125	SF 81	190002	Lode	MGI	05/24/2006	Niagara	08/11/2006	312035	125
126	SF 82	190003	Lode	MGI	05/24/2006	Niagara	08/11/2006	312036	126
127	SF 83	190004	Lode	MGI	05/24/2006	Niagara	08/11/2006	312037	127
128	SF 84	190005	Lode	MGI	05/24/2006	Niagara	08/11/2006	312038	128
129	SF 85	190006	Lode	MGI	05/24/2006	Niagara	08/11/2006	312039	129
130	SF 86	190007	Lode	MGI	05/24/2006	Niagara	08/11/2006	312040	130
131	SF 87	190008	Lode	MGI	05/24/2006	Niagara	08/11/2006	312041	131
132	SF 88	190009	Lode	MGI	05/24/2006	Niagara	08/11/2006	312042	132
133	SF 89	190010	Lode	MGI	05/24/2006	Niagara	08/11/2006	312043	133
134	SF 90	190011	Lode	MGI	05/24/2006	Niagara	08/11/2006	312044	134
135	SF 91	190012	Lode	MGI	05/24/2006	Niagara	08/11/2006	312048	135
136	SF 92	190013	Lode	MGI	05/24/2006	Niagara	08/11/2006	312049	136
137	SF 93	190014	Lode	MGI	05/24/2006	Niagara	08/11/2006	312050	137
138	SF 94	190015	Lode	MGI	05/24/2006	Niagara	08/11/2006	312051	138
139	SF 95	190016	Lode	MGI	05/24/2006	Niagara	08/11/2006	312061	139

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
140	SF 96	190017	Lode	MGI	05/24/2006	Niagara	08/11/2006	312062	140
141	SF 97	190018	Lode	MGI	05/24/2006	Niagara	08/11/2006	312063	141
142	SF 98	190019	Lode	MGI	05/24/2006	Niagara	08/11/2006	312064	142
143	SF 99	190020	Lode	MGI	05/24/2006	Niagara	08/11/2006	312065	143
144	SF 100	190021	Lode	MGI	05/24/2006	Niagara	08/11/2006	312066	144
145	SF 101	199737	Lode	MGI	09/16/2009	MGIAC	11/13/2009	347155	145
146	SF 102	190023	Lode	MGI	05/24/2006	Niagara	08/11/2006	312068	146
147	SF 103	190024	Lode	MGI	05/24/2006	Niagara	08/11/2006	312069	147
148	SF 104	190025	Lode	MGI	05/24/2006	Niagara	08/11/2006	312070	148
149	SF 105	190026	Lode	MGI	05/24/2006	Niagara	08/11/2006	312071	149
150	SF 106	190027	Lode	MGI	05/24/2006	Niagara	08/11/2006	312072	150
151	SF 107	190028	Lode	MGI	05/24/2006	Niagara	08/11/2006	312073	151
152	SF 108	190029	Lode	MGI	05/24/2006	Niagara	08/11/2006	312074	152
153	SF 109	190030	Lode	MGI	05/24/2006	Niagara	08/11/2006	312075	153
154	SF 110	190031	Lode	MGI	05/24/2006	Niagara	08/11/2006	312076	154
155	SF 111	190032	Lode	MGI	05/24/2006	Niagara	08/11/2006	312077	155
156	SF 112	190033	Lode	MGI	05/24/2006	Niagara	08/11/2006	312078	156
157	SF 113	190034	Lode	MGI	05/24/2006	Niagara	08/11/2006	312079	157
158	SF 114	190035	Lode	MGI	05/24/2006	Niagara	08/11/2006	312080	158
159	SF 115	190036	Lode	MGI	05/24/2006	Niagara	08/11/2006	312081	159
160	SF 116	190037	Lode	MGI	05/24/2006	Niagara	08/11/2006	312082	160
161	SF 117	190038	Lode	MGI	05/24/2006	Niagara	08/11/2006	312083	161
162	SF 118	190039	Lode	MGI	05/24/2006	Niagara	08/11/2006	312084	162
163	SF 125	199736	Lode	MGI	09/16/2009	MGIAC	11/13/2009	347156	163
164	SF 126	190041	Lode	MGI	05/24/2006	Niagara	08/11/2006	312086	164
165	SF 127	190042	Lode	MGI	05/24/2006	Niagara	08/11/2006	312087	165
166	SF 128	190043	Lode	MGI	05/24/2006	Niagara	08/11/2006	312088	166
167	SF 129	190044	Lode	MGI	05/24/2006	Niagara	08/11/2006	312089	167

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
168	SF 130	190045	Lode	MGI	05/24/2006	Niagara	08/11/2006	312090	168
169	SF 131	199739	Lode	MGI	09/16/2009	MGIAC	11/13/2009	347157	169
170	SF 132	190047	Lode	MGI	07/27/2006	Niagara	08/11/2006	312072	170
171	SF 133	194738	Lode	MGI	12/11/2007	Niagara	03/03/2008	329605	171
172	SF 134	194739	Lode	MGI	12/11/2007	Niagara	03/03/2008	329606	172
173	SF 135	194740	Lode	MGI	12/11/2007	Niagara	03/03/2008	329607	173
174	SF 136	194741	Lode	MGI	12/11/2007	Niagara	03/03/2008	329608	174
175	SF 137	194742	Lode	MGI	12/11/2007	Niagara	03/03/2008	329609	175
176	SF 138	194743	Lode	MGI	12/11/2007	Niagara	03/03/2008	329610	176
177	SF 139	194744	Lode	MGI	12/11/2007	Niagara	03/03/2008	329611	177
178	SF 140	194745	Lode	MGI	12/11/2007	Niagara	03/03/2008	329612	178
179	SF 141	194746	Lode	MGI	12/11/2007	Niagara	03/03/2008	329613	179
180	SF 142	194747	Lode	MGI	12/11/2007	Niagara	03/03/2008	329614	180
181	SF 143	194748	Lode	MGI	12/11/2007	Niagara	03/03/2008	329615	181
182	SF 144	194749	Lode	MGI	12/11/2007	Niagara	03/03/2008	329616	182
183	SF 145	194750	Lode	MGI	12/11/2007	Niagara	03/03/2008	329617	183
184	SF 146	194751	Lode	MGI	12/11/2007	Niagara	03/03/2008	329618	184
185	SF 147	194752	Lode	MGI	12/11/2007	Niagara	03/03/2008	329619	185
186	SF 148	194753	Lode	MGI	12/11/2007	Niagara	03/03/2008	329620	186
187	SF 149	194754	Lode	MGI	12/11/2007	Niagara	03/03/2008	329621	187
188	SF 150	194755	Lode	MGI	12/11/2007	Niagara	03/03/2008	329622	188
189	SF 151	194756	Lode	MGI	12/11/2007	Niagara	03/03/2008	329623	189
190	SF 152	194757	Lode	MGI	12/11/2007	Niagara	03/03/2008	329624	190
191	SF 153	194758	Lode	MGI	12/11/2007	Niagara	03/03/2008	329625	191
192	SF 154	194759	Lode	MGI	12/11/2007	Niagara	03/03/2008	329626	192
193	SF 155	194760	Lode	MGI	12/11/2007	Niagara	03/03/2008	329627	193
194	SF 156	194761	Lode	MGI	12/11/2007	Niagara	03/03/2008	329628	194
195	SF 157	194762	Lode	MGI	12/11/2007	Niagara	03/03/2008	329629	195

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
196	SF 158	194763	Lode	MGI	12/11/2007	Niagara	03/03/2008	329630	196
197	SF 159	194764	Lode	MGI	12/11/2007	Niagara	03/03/2008	329631	197
198	SF 160	194765	Lode	MGI	12/11/2007	Niagara	03/03/2008	329632	198
199	SF 161	194766	Lode	MGI	12/11/2007	Niagara	03/03/2008	329633	199
200	SF 162	194767	Lode	MGI	12/11/2007	Niagara	03/03/2008	329634	200
201	SF 163	194768	Lode	MGI	12/11/2007	Niagara	03/03/2008	329635	201
202	SF 164	194769	Lode	MGI	12/11/2007	Niagara	03/03/2008	329636	202
203	SF 165	194770	Lode	MGI	12/11/2007	Niagara	03/03/2008	329637	203
204	SF 166	194771	Lode	MGI	12/11/2007	Niagara	03/03/2008	329638	204
205	SF 167	194772	Lode	MGI	12/11/2007	Niagara	03/03/2008	329639	205
206	SF 168	194773	Lode	MGI	12/11/2007	Niagara	03/03/2008	329640	206
207	SF 169	194774	Lode	MGI	12/11/2007	Niagara	03/03/2008	329641	207
208	SF 170	194775	Lode	MGI	12/11/2007	Niagara	03/03/2008	329642	208
209	SF 171	194776	Lode	MGI	12/11/2007	Niagara	03/03/2008	329643	209
210	SF 172	194777	Lode	MGI	12/11/2007	Niagara	03/03/2008	329644	210
211	SF 173	194778	Lode	MGI	12/11/2007	Niagara	03/03/2008	329645	211
212	SF 174	194779	Lode	MGI	12/11/2007	Niagara	03/03/2008	329646	212
213	SF 175	194780	Lode	MGI	12/11/2007	Niagara	03/03/2008	329647	213
214	SF 176	194781	Lode	MGI	12/11/2007	Niagara	03/03/2008	329648	214
215	SF 177	194782	Lode	MGI	12/11/2007	Niagara	03/03/2008	329649	215
216	SF 178	194783	Lode	MGI	12/11/2007	Niagara	03/03/2008	329650	216
217	SF 179	194784	Lode	MGI	12/11/2007	Niagara	03/03/2008	329651	217
218	SF 180	194785	Lode	MGI	12/11/2007	Niagara	03/03/2008	329652	218
219	SF 181	194786	Lode	MGI	12/11/2007	Niagara	03/03/2008	329653	219
220	SF 182	194787	Lode	MGI	12/11/2007	Niagara	03/03/2008	329654	220
221	SF 183	194788	Lode	MGI	12/11/2007	Niagara	03/03/2008	329655	221
222	SF 184	194789	Lode	MGI	12/11/2007	Niagara	03/03/2008	329656	222
223	SF 185	194790	Lode	MGI	12/11/2007	Niagara	03/03/2008	329657	223

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
224	SF 186	194791	Lode	MGI	12/11/2007	Niagara	03/03/2008	329658	224
225	SF 187	194792	Lode	MGI	12/11/2007	Niagara	03/03/2008	329659	225
226	SF 188	194793	Lode	MGI	12/11/2007	Niagara	03/03/2008	329660	226
227	SF 189	194794	Lode	MGI	12/11/2007	Niagara	03/03/2008	329661	227
228	SF 190	194795	Lode	MGI	12/11/2007	Niagara	03/03/2008	329662	228
229	SF 191	194796	Lode	MGI	12/11/2007	Niagara	03/03/2008	329663	229
230	SF 192	199740	Lode	MGI	09/17/2009	MGIAC	11/13/2009	347158	230
231	SF 193	199741	Lode	MGI	09/16/2009	MGIAC	11/13/2009	347159	231
232	SF 194	199742	Lode	MGI	09/16/2009	MGIAC	11/13/2009	347160	232
233	SF 195	199743	Lode	MGI	09/17/2009	MGIAC	11/13/2009	347161	233
234	SF 196	199744	Lode	MGI	09/17/2009	MGIAC	11/13/2009	347162	234
235	SF 197	199745	Lode	MGI	09/17/2009	MGIAC	11/13/2009	347164	235
236	SF 198	199746	Lode	MGI	09/17/2009	MGIAC	11/13/2009	347163	236
237	SF 199	199747	Lode	MGI	09/17/2009	MGIAC	11/13/2009	347165	237
238	SF 200	199748	Lode	MGI	09/17/2009	MGIAC	11/13/2009	347166	238
239	SF 201	199749	Lode	MGI	09/17/2009	MGIAC	11/13/2009	347167	239
240	SF 202	199750	Lode	MGI	09/17/2009	MGIAC	11/13/2009	347168	240
241	SF 203	199751	Lode	MGI	09/16/2009	MGIAC	11/13/2009	347169	241
242	SF 204	199752	Lode	MGI	09/16/2009	MGIAC	11/13/2009	347170	242
243	SF 205	199753	Lode	MGI	09/18/2009	MGIAC	11/13/2009	347171	243
244	SF 206	199754	Lode	MGI	09/18/2009	MGIAC	11/13/2009	347172	244
245	SF 207	199755	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347173	245
246	SF 208	199756	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347174	246
247	SF 209	199757	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347175	247
248	SF 210	199758	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347176	248
249	SF 211	199759	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347177	249
250	SF 212	199760	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347178	250
251	SF 213	199761	Lode	MGI	09/18/2009	MGIAC	11/13/2009	347179	251

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
252	SF 214	199762	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347180	252
253	SF 215	199763	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347186	253
254	SF 216	199764	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347187	254
255	SF 217	199765	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347188	255
256	SF 218	199766	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347189	256
257	SF 219	199767	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347190	257
258	SF 220	199768	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347194	258
259	SF 221	199769	Lode	MGI	09/19/2009	MGIAC	11/13/2009	347195	259
260	SF 222	199770	Lode	MGI	09/18/2009	MGIAC	11/13/2009	347196	260
261	SF 223	200326	Lode	MGI	01/12/2001	MGI	02/16/2010	349501	261
262	SF 224	200327	Lode	MGI	01/12/2001	MGI	02/16/2010	349497	262
263	SF 225	200328	Lode	MGI	01/12/2001	MGI	02/16/2010	349500	263
264	SF 226	200329	Lode	MGI	01/12/2001	MGI	02/16/2010	349502	264
265	SF 227	200330	Lode	MGI	01/12/2001	MGI	02/16/2010	349503	265
266	SF 228	200331	Lode	MGI	01/12/2001	MGI	02/16/2010	349504	266
267	SF 229	200332	Lode	MGI	01/12/2001	MGI	02/16/2010	349505	267
268	SF 230	200333	Lode	MGI	01/12/2001	MGI	02/16/2010	349506	268
269	SF 231	200334	Lode	MGI	01/12/2001	MGI	02/16/2010	349507	269
270	SF 232	200335	Lode	MGI	01/12/2001	MGI	02/16/2010	349508	270
271	SF 233	200336	Lode	MGI	01/12/2001	MGI	02/16/2010	349509	271
272	SF 234	200337	Lode	MGI	01/12/2001	MGI	02/16/2010	349510	272
273	SF 235	201078	Lode	MGI	04/20/2010	MGI	06/11/2010	352443	273
274	SF 236	201079	Lode	MGI	04/20/2010	MGI	06/11/2010	352444	274
275	SF 237	201080	Lode	MGI	04/20/2010	MGI	06/11/2010	352445	275
276	SF 238	201081	Lode	MGI	04/20/2010	MGI	06/11/2010	352446	276
277	SF 239	201082	Lode	MGI	04/20/2010	MGI	06/11/2010	352447	277
278	SF 240	201083	Lode	MGI	04/20/2010	MGI	06/11/2010	352448	278
279	SF 241	201084	Lode	MGI	04/20/2010	MGI	06/11/2010	352449	279

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
280	SF 242	201085	Lode	MGI	04/20/2010	MGI	06/11/2010	352450	280
281	SF 243	201086	Lode	MGI	04/20/2010	MGI	06/11/2010	352451	281
282	SF 244	201087	Lode	MGI	04/20/2010	MGI	06/11/2010	352452	282
283	SF 245	201088	Lode	MGI	04/20/2010	MGI	06/11/2010	352453	283
284	SF 246	201089	Lode	MGI	04/20/2010	MGI	06/11/2010	352454	284
285	SF 247	201090	Lode	MGI	04/20/2010	MGI	06/11/2010	352457	285
286	SF 248	201091	Lode	MGI	04/20/2010	MGI	06/11/2010	352458	286
287	SF 249	201092	Lode	MGI	04/20/2010	MGI	06/11/2010	352459	287
288	SF 250	201093	Lode	MGI	04/20/2010	MGI	06/11/2010	352460	288
289	SF 251	201094	Lode	MGI	04/20/2010	MGI	06/11/2010	352461	289
290	SF 252	201095	Lode	MGI	04/20/2010	MGI	06/11/2010	352462	290
291	SF 253	201096	Lode	MGI	04/20/2010	MGI	06/11/2010	352463	291
292	SF 254	201097	Lode	MGI	04/20/2010	MGI	06/11/2010	352464	292
293	SF 255	201098	Lode	MGI	04/20/2010	MGI	06/11/2010	352389	293
294	SF 256	201099	Lode	MGI	04/20/2010	MGI	06/11/2010	352390	294
295	SF 257	201100	Lode	MGI	04/20/2010	MGI	06/11/2010	352392	295
296	SF 258	201101	Lode	MGI	04/20/2010	MGI	06/11/2010	352393	296
297	SF 259	201102	Lode	MGI	04/20/2010	MGI	06/11/2010	352394	297
298	SF 260	201103	Lode	MGI	04/20/2010	MGI	06/11/2010	352395	298
299	SF 261	201104	Lode	MGI	04/20/2010	MGI	06/11/2010	352397	299
300	SF 262	201105	Lode	MGI	04/20/2010	MGI	06/11/2010	352399	300
301	SF 263	201106	Lode	MGI	04/20/2010	MGI	06/11/2010	352401	301
302	SF 264	201107	Lode	MGI	04/20/2010	MGI	06/11/2010	352403	302
303	SF 265	201108	Lode	MGI	04/20/2010	MGI	06/11/2010	352405	303
304	SF 266	201109	Lode	MGI	04/20/2010	MGI	06/11/2010	352430	304
305	SF 267	201110	Lode	MGI	04/20/2010	MGI	06/11/2010	352431	305
306	SF 268	201111	Lode	MGI	04/20/2010	MGI	06/11/2010	352436	306
307	SF 269	201112	Lode	MGI	04/20/2010	MGI	06/11/2010	352437	307

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
308	SF 270	201113	Lode	MGI	04/20/2010	MGI	06/11/2010	352438	308
309	SF 271	201114	Lode	MGI	04/20/2010	MGI	06/11/2010	352439	309
310	SF 272	201115	Lode	MGI	04/20/2010	MGI	06/11/2010	352440	310
311	SF 273	201116	Lode	MGI	04/20/2010	MGI	06/11/2010	352441	311
312	SF 274	201117	Lode	MGI	04/20/2010	MGI	06/11/2010	352442	312
313	SF 275	201118	Lode	MGI	04/20/2010	MGI	06/11/2010	352443	313
314	SF 276	201119	Lode	MGI	04/20/2010	MGI	06/11/2010	352345	314
315	SF 277	201120	Lode	MGI	04/20/2010	MGI	06/11/2010	352351	315
316	SF 278	201121	Lode	MGI	04/20/2010	MGI	06/11/2010	352354	316
317	SF 279	201122	Lode	MGI	04/20/2010	MGI	06/11/2010	352355	317
318	SF 280	201123	Lode	MGI	04/20/2010	MGI	06/11/2010	352356	318
319	SF 281	201124	Lode	MGI	04/20/2010	MGI	06/11/2010	352357	319
320	SF 282	201125	Lode	MGI	04/20/2010	MGI	06/11/2010	352358	320
321	SF 283	201126	Lode	MGI	04/20/2010	MGI	06/11/2010	352359	321
322	SF 284	201127	Lode	MGI	04/20/2010	MGI	06/11/2010	352368	322
323	SF 285	201128	Lode	MGI	04/20/2010	MGI	06/11/2010	352370	323
324	SF 286	201129	Lode	MGI	04/20/2010	MGI	06/11/2010	352373	324
325	SF 287	201130	Lode	MGI	04/20/2010	MGI	06/11/2010	352375	325
326	SF 288	201131	Lode	MGI	04/20/2010	MGI	06/11/2010	352377	326
327	SF 289	201132	Lode	MGI	04/20/2010	MGI	06/11/2010	352379	327
328	SF 290	201133	Lode	MGI	04/20/2010	MGI	06/11/2010	352380	328
329	SF 291	201134	Lode	MGI	04/20/2010	MGI	06/11/2010	352381	329
330	SF 292	201135	Lode	MGI	04/20/2010	MGI	06/11/2010	352382	330
331	SF 293	201136	Lode	MGI	04/20/2010	MGI	06/11/2010	352384	331
332	SF 294	201137	Lode	MGI	04/20/2010	MGI	06/11/2010	352387	332
333	SF 295	201138	Lode	MGI	04/20/2010	MGI	06/11/2010	352413	333
334	SF 296	201139	Lode	MGI	04/20/2010	MGI	06/11/2010	352414	334
335	SF 297	201140	Lode	MGI	04/20/2010	MGI	06/11/2010	352415	335

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
336	SF 298	201141	Lode	MGI	04/20/2010	MGI	06/11/2010	352416	336
337	SF 299	201142	Lode	MGI	04/20/2010	MGI	06/11/2010	352417	337
338	SF 300	201143	Lode	MGI	04/20/2010	MGI	06/11/2010	352418	338
339	SF 301	201144	Lode	MGI	04/20/2010	MGI	06/11/2010	352419	339
340	SF 302	201145	Lode	MGI	04/20/2010	MGI	06/11/2010	352420	340
341	SF 303	201146	Lode	MGI	04/20/2010	MGI	06/11/2010	352421	341
342	SF 304	201147	Lode	MGI	04/20/2010	MGI	06/11/2010	352422	342
343	SF 305	201148	Lode	MGI	04/20/2010	MGI	06/11/2010	352423	343
344	SF 306	201149	Lode	MGI	04/20/2010	MGI	06/11/2010	352424	344
345	SF 307	201150	Lode	MGI	04/20/2010	MGI	06/11/2010	352425	345
346	SF 308	201151	Lode	MGI	04/20/2010	MGI	06/11/2010	352426	346
347	SF 309	201152	Lode	MGI	04/20/2010	MGI	06/11/2010	352427	347
348	SF 310	201153	Lode	MGI	04/20/2010	MGI	06/11/2010	352428	348
349	SF 311	201154	Lode	MGI	04/20/2010	MGI	06/11/2010	352429	349
350	SF 312	201155	Lode	MGI	04/20/2010	MGI	06/11/2010	352336	350
351	SF 313	201156	Lode	MGI	04/20/2010	MGI	06/11/2010	352339	351
352	SF 314	201157	Lode	MGI	04/20/2010	MGI	06/11/2010	352341	352
353	SF 315	201158	Lode	MGI	04/20/2010	MGI	06/11/2010	352374	353
354	SF 316	201159	Lode	MGI	04/20/2010	MGI	06/11/2010	352376	354
355	SF 317	201160	Lode	MGI	04/20/2010	MGI	06/11/2010	352378	355
356	SF 318	201161	Lode	MGI	04/20/2010	MGI	06/11/2010	352383	356
357	SF 319	201162	Lode	MGI	04/20/2010	MGI	06/11/2010	352385	357
358	SF 320	201163	Lode	MGI	04/20/2010	MGI	06/11/2010	352386	358
359	SF 321	201164	Lode	MGI	04/20/2010	MGI	06/11/2010	352388	359
360	SF 322	201165	Lode	MGI	04/20/2010	MGI	06/11/2010	352391	360
361	SF 323	201166	Lode	MGI	04/20/2010	MGI	06/11/2010	352396	361
362	SF 324	201167	Lode	MGI	04/20/2010	MGI	06/11/2010	352398	362
363	SF 325	201168	Lode	MGI	04/20/2010	MGI	06/11/2010	352400	363

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
364	SF 326	201169	Lode	MGI	04/20/2010	MGI	06/11/2010	352402	364
365	SF 327	201170	Lode	MGI	04/20/2010	MGI	06/11/2010	352404	365
366	SF 328	201171	Lode	MGI	04/20/2010	MGI	06/11/2010	352406	366
367	SF 329	201172	Lode	MGI	04/20/2010	MGI	06/11/2010	352407	367
368	SF 330	201173	Lode	MGI	04/20/2010	MGI	06/11/2010	352408	368
369	SF 331	201174	Lode	MGI	04/20/2010	MGI	06/11/2010	352409	369
370	SF 332	201175	Lode	MGI	04/20/2010	MGI	06/11/2010	352410	370
371	SF 333	201176	Lode	MGI	04/20/2010	MGI	06/11/2010	352411	371
372	SF 334	201177	Lode	MGI	04/20/2010	MGI	06/11/2010	352412	372
373	SF 335	201178	Lode	MGI	04/20/2010	MGI	06/11/2010	352333	373
374	SF 336	201179	Lode	MGI	04/20/2010	MGI	06/11/2010	352334	374
375	SF 337	201180	Lode	MGI	04/20/2010	MGI	06/11/2010	352335	375
376	SF 338	201181	Lode	MGI	04/20/2010	MGI	06/11/2010	352337	376
377	SF 339	201182	Lode	MGI	04/20/2010	MGI	06/11/2010	352338	377
378	SF 340	201183	Lode	MGI	04/20/2010	MGI	06/11/2010	352340	378
379	SF 341	201184	Lode	MGI	04/20/2010	MGI	06/11/2010	352342	379
380	SF 342	201185	Lode	MGI	04/20/2010	MGI	06/11/2010	352344	380
381	SF 343	201186	Lode	MGI	04/20/2010	MGI	06/11/2010	352346	381
382	SF 344	201187	Lode	MGI	04/20/2010	MGI	06/11/2010	352347	382
383	SF 345	201188	Lode	MGI	04/20/2010	MGI	06/11/2010	352348	383
384	SF 346	201189	Lode	MGI	04/20/2010	MGI	06/11/2010	352349	384
385	SF 347	201190	Lode	MGI	04/20/2010	MGI	06/11/2010	352350	385
386	SF 348	201191	Lode	MGI	04/20/2010	MGI	06/11/2010	352352	386
387	SF 349	201192	Lode	MGI	04/20/2010	MGI	06/11/2010	352353	387
388	SF 350	201193	Lode	MGI	04/20/2010	MGI	06/11/2010	352360	388
389	SF 351	201194	Lode	MGI	04/20/2010	MGI	06/11/2010	352367	389
390	SF 352	201195	Lode	MGI	04/20/2010	MGI	06/11/2010	352369	390
391	SF 353	201196	Lode	MGI	04/20/2010	MGI	06/11/2010	352371	391

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
392	SF 354	201197	Lode	MGI	04/20/2010	MGI	06/11/2010	352372	392
393	SF 355	201198	Lode	MGI	04/20/2010	MGI	06/11/2010	352302	393
394	SF 356	201199	Lode	MGI	04/20/2010	MGI	06/11/2010	352303	394
395	SF 357	201200	Lode	MGI	04/20/2010	MGI	06/11/2010	352304	395
396	SF 358	201201	Lode	MGI	04/20/2010	MGI	06/11/2010	352312	396
397	SF 359	201202	Lode	MGI	04/20/2010	MGI	06/11/2010	352313	397
398	SF 360	201203	Lode	MGI	04/20/2010	MGI	06/11/2010	352314	398
399	SF 361	201204	Lode	MGI	04/20/2010	MGI	06/11/2010	352315	399
400	SF 362	201205	Lode	MGI	04/20/2010	MGI	06/11/2010	352316	400
401	SF 363	201206	Lode	MGI	04/20/2010	MGI	06/11/2010	352317	401
402	SF 364	201207	Lode	MGI	04/20/2010	MGI	06/11/2010	352318	402
403	SF 365	201208	Lode	MGI	04/20/2010	MGI	06/11/2010	352319	403
404	SF 366	201209	Lode	MGI	04/21/2010	MGI	06/11/2010	352320	404
405	SF 367	201210	Lode	MGI	04/21/2010	MGI	06/11/2010	352321	405
406	SF 368	201211	Lode	MGI	04/21/2010	MGI	06/11/2010	352322	406
407	SF 369	201212	Lode	MGI	04/21/2010	MGI	06/11/2010	352323	407
408	SF 370	201213	Lode	MGI	04/21/2010	MGI	06/11/2010	352324	408
409	SF 371	201214	Lode	MGI	04/21/2010	MGI	06/11/2010	352325	409
410	SF 372	201215	Lode	MGI	04/21/2010	MGI	06/11/2010	352326	410
411	SF 373	201216	Lode	MGI	04/21/2010	MGI	06/11/2010	352331	411
412	SF 374	201217	Lode	MGI	04/21/2010	MGI	06/11/2010	352332	412
413	SF 375	201218	Lode	MGI	04/21/2010	MGI	06/11/2010	352275	413
414	SF 376	201219	Lode	MGI	04/21/2010	MGI	06/11/2010	352276	414
415	SF 377	201220	Lode	MGI	04/21/2010	MGI	06/11/2010	352277	415
416	SF 378	201221	Lode	MGI	04/21/2010	MGI	06/11/2010	352278	416
417	SF 379	201222	Lode	MGI	04/21/2010	MGI	06/11/2010	352279	417
418	SF 380	201223	Lode	MGI	04/21/2010	MGI	06/11/2010	352280	418
419	SF 381	201224	Lode	MGI	04/21/2010	MGI	06/11/2010	352281	419

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
420	SF 382	201225	Lode	MGI	04/21/2010	MGI	06/11/2010	352282	420
421	SF 383	201226	Lode	MGI	04/21/2010	MGI	06/11/2010	352283	421
422	SF 384	201227	Lode	MGI	04/21/2010	MGI	06/11/2010	352287	422
423	SF 385	201228	Lode	MGI	04/21/2010	MGI	06/11/2010	352288	423
424	SF 386	201229	Lode	MGI	04/21/2010	MGI	06/11/2010	352289	424
425	SF 387	201230	Lode	MGI	04/21/2010	MGI	06/11/2010	352293	425
426	SF 388	201231	Lode	MGI	04/21/2010	MGI	06/11/2010	352294	426
427	SF 389	201232	Lode	MGI	04/21/2010	MGI	06/11/2010	352295	427
428	SF 390	201233	Lode	MGI	04/21/2010	MGI	06/11/2010	352296	428
429	SF 391	201234	Lode	MGI	04/21/2010	MGI	06/11/2010	352298	429
430	SF 392	201235	Lode	MGI	04/21/2010	MGI	06/11/2010	352299	430
431	SF 393	201236	Lode	MGI	04/21/2010	MGI	06/11/2010	352300	431
432	SF 394	201237	Lode	MGI	04/21/2010	MGI	06/11/2010	352301	432
433	SF 395	201238	Lode	MGI	04/21/2010	MGI	06/11/2010	352252	433
434	SF 396	201239	Lode	MGI	04/21/2010	MGI	06/11/2010	352253	434
435	SF 397	201240	Lode	MGI	04/21/2010	MGI	06/11/2010	352254	435
436	SF 398	201241	Lode	MGI	04/21/2010	MGI	06/11/2010	352255	436
437	SF 399	201242	Lode	MGI	04/21/2010	MGI	06/11/2010	352257	437
438	SF 400	201243	Lode	MGI	04/21/2010	MGI	06/11/2010	352258	438
439	SF 401	201244	Lode	MGI	04/21/2010	MGI	06/11/2010	352259	439
440	SF 402	201245	Lode	MGI	04/21/2010	MGI	06/11/2010	352260	440
441	SF 403	201246	Lode	MGI	04/21/2010	MGI	06/11/2010	352261	441
442	SF 404	201247	Lode	MGI	04/21/2010	MGI	06/11/2010	352263	442
443	SF 405	201248	Lode	MGI	04/21/2010	MGI	06/11/2010	352264	443
444	SF 406	201249	Lode	MGI	04/21/2010	MGI	06/11/2010	352265	444
445	SF 407	201250	Lode	MGI	04/21/2010	MGI	06/11/2010	352270	445
446	SF 408	201251	Lode	MGI	04/21/2010	MGI	06/11/2010	352271	446
447	SF 409	201252	Lode	MGI	04/21/2010	MGI	06/11/2010	352272	447

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
448	SF 410	201253	Lode	MGI	04/21/2010	MGI	06/11/2010	352273	448
449	SF 411	201254	Lode	MGI	04/21/2010	MGI	06/11/2010	352274	449
450	SF 412	203035	Lode	MGI	10/28/2010	MGI	11/23/2010	356587	450
451	SF 413	203036	Lode	MGI	10/28/2010	MGI	11/23/2010	356589	451
452	SF 414	203037	Lode	MGI	10/28/2010	MGI	11/23/2010	356590	452
453	SF 415	203038	Lode	MGI	10/28/2010	MGI	11/23/2010	356591	453
454	SF 416	203039	Lode	MGI	10/28/2010	MGI	11/23/2010	356592	454
455	SF 417	203040	Lode	MGI	10/28/2010	MGI	11/23/2010	356593	455
456	SF 418	203041	Lode	MGI	10/28/2010	MGI	11/23/2010	356594	456
457	SF 419	203042	Lode	MGI	10/28/2010	MGI	11/23/2010	356595	457
458	SF 420	203043	Lode	MGI	10/28/2010	MGI	11/23/2010	356596	458
459	SF 421	203044	Lode	MGI	10/28/2010	MGI	11/23/2010	356597	459
460	SF 422	203045	Lode	MGI	10/28/2010	MGI	11/23/2010	356598	460
461	SF 423	203046	Lode	MGI	10/28/2010	MGI	11/23/2010	356599	461
462	SF 424	203047	Lode	MGI	10/28/2010	MGI	11/23/2010	356600	462
463	SF 425	203048	Lode	MGI	10/28/2010	MGI	11/23/2010	356614	463
464	SF 426	203049	Lode	MGI	10/28/2010	MGI	11/23/2010	356617	464
465	SF 427	203050	Lode	MGI	10/28/2010	MGI	11/23/2010	356619	465
466	SF 428	203051	Lode	MGI	10/28/2010	MGI	11/23/2010	356620	466
467	SF 429	203052	Lode	MGI	10/28/2010	MGI	11/23/2010	356621	467
468	SF 430	203053	Lode	MGI	10/28/2010	MGI	11/23/2010	356622	468
469	SF 431	203054	Lode	MGI	10/28/2010	MGI	11/23/2010	356608	469
470	SF 432	203055	Lode	MGI	10/28/2010	MGI	11/23/2010	356609	470
471	SF 433	203056	Lode	MGI	10/28/2010	MGI	11/23/2010	356610	471
472	SF 434	203057	Lode	MGI	10/28/2010	MGI	11/23/2010	356611	472
473	SF 435	203058	Lode	MGI	10/28/2010	MGI	11/23/2010	356612	473
474	SF 436	203059	Lode	MGI	10/28/2010	MGI	11/23/2010	356613	474
475	SF 437	203060	Lode	MGI	10/28/2010	MGI	11/23/2010	356615	475

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
476	SF 438	203061	Lode	MGI	10/28/2010	MGI	11/23/2010	356616	476
477	SF 439	203062	Lode	MGI	10/28/2010	MGI	11/23/2010	356618	477
478	SF 451	205314	Lode	MGI	06/20/2011	MGI	07/05/2011	361283	478
479	SF 452	205315	Lode	MGI	06/20/2011	MGI	07/05/2011	361282	479
480	SF 453 A	211429	Lode	MGI	08/18/2012	MGI	09/06/2012	371928	480
481	SF 456	206796	Lode	MGI	08/23/2011	MGI	11/08/2011	364063	481
482	SF 457	206797	Lode	MGI	08/23/2011	MGI	11/08/2011	364064	482
483	SF 458	206798	Lode	MGI	08/23/2011	MGI	11/08/2011	364065	483
484	SF 459	206799	Lode	MGI	08/23/2011	MGI	11/08/2011	364066	484
485	SF 460	206800	Lode	MGI	08/23/2011	MGI	11/08/2011	364067	485
486	SF 461	206801	Lode	MGI	08/23/2011	MGI	11/08/2011	364068	486
487	SF 462	206802	Lode	MGI	08/23/2011	MGI	11/08/2011	364069	487
488	SF 463	206803	Lode	MGI	08/23/2011	MGI	11/08/2011	364070	488
489	SF 464	206804	Lode	MGI	08/23/2011	MGI	11/08/2011	364071	489
490	SF 465	206805	Lode	MGI	08/23/2011	MGI	11/08/2011	364072	490
491	SF 466	206806	Lode	MGI	08/23/2011	MGI	11/08/2011	364073	491
492	SF 467	206807	Lode	MGI	08/23/2011	MGI	11/08/2011	364077	492
493	SF 468	206808	Lode	MGI	08/23/2011	MGI	11/08/2011	364078	493
494	SF 469	206809	Lode	MGI	08/23/2011	MGI	11/08/2011	364079	494
495	SF 470	206810	Lode	MGI	08/23/2011	MGI	11/08/2011	364080	495
496	SF 471	206811	Lode	MGI	08/23/2011	MGI	11/08/2011	364081	496
497	SF 472	206812	Lode	MGI	08/23/2011	MGI	11/08/2011	364082	497
498	SF 473	206813	Lode	MGI	08/23/2011	MGI	11/08/2011	364083	498
499	SF 474	206814	Lode	MGI	08/23/2011	MGI	11/08/2011	364084	499
500	SF 475	206815	Lode	MGI	08/23/2011	MGI	11/08/2011	364085	500
501	SF 476	206816	Lode	MGI	08/23/2011	MGI	11/08/2011	364086	501
502	SF 477	206817	Lode	MGI	08/23/2011	MGI	11/08/2011	364087	502
503	SF 478	206818	Lode	MGI	08/23/2011	MGI	11/08/2011	364088	503

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
504	SF 479	206819	Lode	MGI	08/23/2011	MGI	11/08/2011	364089	504
505	SF 480	206820	Lode	MGI	08/23/2011	MGI	11/08/2011	364090	505
506	SF 481	206821	Lode	MGI	08/23/2011	MGI	11/08/2011	364091	506
507	SF 482	206822	Lode	MGI	08/23/2011	MGI	11/08/2011	364092	507
508	SF 483	206823	Lode	MGI	08/23/2011	MGI	11/08/2011	364093	508
509	SF 484	206824	Lode	MGI	08/23/2011	MGI	11/08/2011	364094	509
510	SF 485	206825	Lode	MGI	08/23/2011	MGI	11/08/2011	364095	510
511	SF 486	206826	Lode	MGI	08/23/2011	MGI	11/08/2011	364096	511
512	SF 487	206827	Lode	MGI	08/23/2011	MGI	11/08/2011	364097	512
513	SF 488	206828	Lode	MGI	08/24/2011	MGI	11/08/2011	364098	513
514	SF 489	206829	Lode	MGI	08/24/2011	MGI	11/08/2011	364099	514
515	SF 490	206830	Lode	MGI	08/24/2011	MGI	11/08/2011	364100	515
516	SF 491	206831	Lode	MGI	08/24/2011	MGI	11/08/2011	364101	516
517	SF 492	206832	Lode	MGI	08/24/2011	MGI	11/08/2011	364102	517
518	SF 493	206833	Lode	MGI	08/24/2011	MGI	11/08/2011	364103	518
519	SF 494	206834	Lode	MGI	08/24/2011	MGI	11/08/2011	364104	519
520	SF 495	206835	Lode	MGI	08/24/2011	MGI	11/08/2011	364105	520
521	SF 496	206836	Lode	MGI	08/24/2011	MGI	11/08/2011	364106	521
522	SF 497	206837	Lode	MGI	08/24/2011	MGI	11/08/2011	364107	522
523	SF 498	206838	Lode	MGI	08/24/2011	MGI	11/08/2011	364108	523
524	SF 499	206839	Lode	MGI	08/24/2011	MGI	11/08/2011	364109	524
525	SF 500	206840	Lode	MGI	08/24/2011	MGI	11/08/2011	364112	525
526	SF 501	206841	Lode	MGI	08/24/2011	MGI	11/08/2011	364113	526
527	SF 502	206842	Lode	MGI	08/24/2011	MGI	11/08/2011	364114	527
528	SF 503	206843	Lode	MGI	08/24/2011	MGI	11/08/2011	364115	528
529	SF 504	206844	Lode	MGI	08/24/2011	MGI	11/08/2011	364116	529
530	SF 505	206845	Lode	MGI	08/24/2011	MGI	11/08/2011	364117	530
531	SF 506	206846	Lode	MGI	08/24/2011	MGI	11/08/2011	364118	531

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
532	SF 507	206847	Lode	MGI	08/24/2011	MGI	11/08/2011	364121	532
533	SF 508	206848	Lode	MGI	08/24/2011	MGI	11/08/2011	364122	533
534	SF 509	206849	Lode	MGI	08/24/2011	MGI	11/08/2011	364123	534
535	SF 510	206850	Lode	MGI	08/24/2011	MGI	11/08/2011	364124	535
536	SF 511	206851	Lode	MGI	08/24/2011	MGI	11/08/2011	364125	536
537	SF 512	206852	Lode	MGI	08/24/2011	MGI	11/08/2011	364126	537
538	SF 513	206853	Lode	MGI	08/24/2011	MGI	11/08/2011	364127	538
539	SF 514	206854	Lode	MGI	08/27/2011	MGI	11/08/2011	364128	539
540	SF 515	206855	Lode	MGI	08/27/2011	MGI	11/08/2011	364129	540
541	SF 516	206856	Lode	MGI	08/27/2011	MGI	11/08/2011	364130	541
542	SF 517	206857	Lode	MGI	08/27/2011	MGI	11/08/2011	364131	542
543	SF 518	206858	Lode	MGI	08/27/2011	MGI	11/08/2011	364132	543
544	SF 519	206859	Lode	MGI	08/27/2011	MGI	11/08/2011	364133	544
545	SF 520	206860	Lode	MGI	08/27/2011	MGI	11/08/2011	364134	545
546	SF 521	206861	Lode	MGI	08/24/2011	MGI	11/08/2011	364135	546
547	SF 522	206862	Lode	MGI	08/24/2011	MGI	11/08/2011	364136	547
548	SF 523	206863	Lode	MGI	08/24/2011	MGI	11/08/2011	364137	548
549	SF 524	206864	Lode	MGI	08/24/2011	MGI	11/08/2011	364138	549
550	SF 525	206865	Lode	MGI	08/24/2011	MGI	11/08/2011	364139	550
551	SF 526	206866	Lode	MGI	08/24/2011	MGI	11/08/2011	364140	551
552	SF 527	206867	Lode	MGI	08/24/2011	MGI	11/08/2011	364142	552
553	SF 528	206868	Lode	MGI	08/24/2011	MGI	11/08/2011	364143	553
554	SF 529	206869	Lode	MGI	08/24/2011	MGI	11/08/2011	364144	554
555	SF 530	206870	Lode	MGI	08/24/2011	MGI	11/08/2011	364145	555
556	SF 531	206871	Lode	MGI	08/24/2011	MGI	11/08/2011	364146	556
557	SF 532	206872	Lode	MGI	08/24/2011	MGI	11/08/2011	364147	557
558	SF 533	206873	Lode	MGI	08/24/2011	MGI	11/08/2011	364149	558
559	SF 534	206874	Lode	MGI	08/24/2011	MGI	11/08/2011	364150	559

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
560	SF 535	206875	Lode	MGI	08/24/2011	MGI	11/08/2011	364152	560
561	SF 536	206876	Lode	MGI	08/24/2011	MGI	11/08/2011	364153	561
562	SF 537	206877	Lode	MGI	08/24/2011	MGI	11/08/2011	364156	562
563	SF 538	206878	Lode	MGI	08/24/2011	MGI	11/08/2011	364157	563
564	SF 539	206879	Lode	MGI	08/24/2011	MGI	11/08/2011	364158	564
565	SF 540	206880	Lode	MGI	08/24/2011	MGI	11/08/2011	364159	565
566	SF 541	206881	Lode	MGI	08/24/2011	MGI	11/08/2011	364160	566
567	SF 542	206882	Lode	MGI	08/24/2011	MGI	11/08/2011	364161	567
568	SF 543	206883	Lode	MGI	08/24/2011	MGI	11/08/2011	364162	568
569	SF 544	206884	Lode	MGI	08/24/2011	MGI	11/08/2011	364163	569
570	SF 545	206885	Lode	MGI	08/24/2011	MGI	11/08/2011	364164	570
571	SF 546	206886	Lode	MGI	08/24/2011	MGI	11/08/2011	364165	571
572	SF 547	206887	Lode	MGI	08/24/2011	MGI	11/08/2011	364166	572
573	SF 548	206888	Lode	MGI	08/24/2011	MGI	11/08/2011	364167	573
574	SF 549	206889	Lode	MGI	08/24/2011	MGI	11/08/2011	364168	574
575	SF 550	206890	Lode	MGI	08/24/2011	MGI	11/08/2011	364169	575
576	SF 551	206891	Lode	MGI	08/24/2011	MGI	11/08/2011	364170	576
577	SF 552	206892	Lode	MGI	08/24/2011	MGI	11/08/2011	364171	577
578	SF 553	206893	Lode	MGI	08/24/2011	MGI	11/08/2011	364172	578
579	SF 554	206894	Lode	MGI	08/24/2011	MGI	11/08/2011	364173	579
580	SF 555	206895	Lode	MGI	08/24/2011	MGI	11/08/2011	364174	580
581	SF 556	206896	Lode	MGI	08/24/2011	MGI	11/08/2011	364175	581
582	SF 557	206897	Lode	MGI	08/24/2011	MGI	11/08/2011	364176	582
583	SF 558	206898	Lode	MGI	08/24/2011	MGI	11/08/2011	364177	583
584	SF 559	206899	Lode	MGI	08/24/2011	MGI	11/08/2011	364178	584
585	SF 560	206900	Lode	MGI	08/24/2011	MGI	11/08/2011	364179	585
586	SF 561	206901	Lode	MGI	08/24/2011	MGI	11/08/2011	364180	586
587	SF 562	206902	Lode	MGI	08/24/2011	MGI	11/08/2011	364181	587

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
588	SF 563	206903	Lode	MGI	08/24/2011	MGI	11/08/2011	364182	588
589	SF 564	206904	Lode	MGI	08/24/2011	MGI	11/08/2011	364183	589
590	SF 565	206905	Lode	MGI	08/24/2011	MGI	11/08/2011	364184	590
591	SF 566	206906	Lode	MGI	08/24/2011	MGI	11/08/2011	364185	591
592	SF 567	206907	Lode	MGI	08/24/2011	MGI	11/08/2011	364186	592
593	SF 568	206908	Lode	MGI	08/24/2011	MGI	11/08/2011	364187	593
594	SF 569	206909	Lode	MGI	08/24/2011	MGI	11/08/2011	364188	594
595	SF 570	206910	Lode	MGI	08/24/2011	MGI	11/08/2011	364189	595
596	SF 571	206911	Lode	MGI	08/24/2011	MGI	11/08/2011	364190	596
597	SF 572	206912	Lode	MGI	08/24/2011	MGI	11/08/2011	364191	597
598	SF 573	206913	Lode	MGI	08/24/2011	MGI	11/08/2011	364192	598
599	SF 574	206914	Lode	MGI	08/24/2011	MGI	11/08/2011	364193	599
600	SF 575	206915	Lode	MGI	08/24/2011	MGI	11/08/2011	364194	600
601	SF 576	206916	Lode	MGI	08/24/2011	MGI	11/08/2011	364195	601
602	SF 577	206917	Lode	MGI	08/24/2011	MGI	11/08/2011	364196	602
603	SF 578	206918	Lode	MGI	08/24/2011	MGI	11/08/2011	364197	603
604	SF 579	206919	Lode	MGI	08/24/2011	MGI	11/08/2011	364198	604
605	SF 580	206920	Lode	MGI	08/24/2011	MGI	11/08/2011	364199	605
606	SF 581	206921	Lode	MGI	08/24/2011	MGI	11/08/2011	364200	606
607	SF 582	206922	Lode	MGI	08/24/2011	MGI	11/08/2011	364201	607
608	SF 583	206923	Lode	MGI	08/24/2011	MGI	11/08/2011	364202	608
609	SF 584	206924	Lode	MGI	08/24/2011	MGI	11/08/2011	364203	609
610	SF 585	206925	Lode	MGI	08/24/2011	MGI	11/08/2011	364204	610
611	SF 586	206926	Lode	MGI	08/24/2011	MGI	11/08/2011	364205	611
612	SF 587	206927	Lode	MGI	08/24/2011	MGI	11/08/2011	364206	612
613	SF 588	206928	Lode	MGI	08/24/2011	MGI	11/08/2011	364207	613
614	SF 589	206929	Lode	MGI	08/24/2011	MGI	11/08/2011	364212	614
615	SF 590	206930	Lode	MGI	08/24/2011	MGI	11/08/2011	364213	615

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
616	SF 591	206931	Lode	MGI	08/24/2011	MGI	11/08/2011	364214	616
617	SF 592	206932	Lode	MGI	08/24/2011	MGI	11/08/2011	364215	617
618	SF 593	206933	Lode	MGI	08/26/2011	MGI	11/08/2011	364216	618
619	SF 594	206934	Lode	MGI	08/26/2011	MGI	11/08/2011	364217	619
620	SF 595	206935	Lode	MGI	08/26/2011	MGI	11/08/2011	364218	620
621	SF 596	206936	Lode	MGI	08/26/2011	MGI	11/08/2011	364219	621
622	SF 597	206937	Lode	MGI	08/26/2011	MGI	11/08/2011	364220	622
623	SF 598	206938	Lode	MGI	08/26/2011	MGI	11/08/2011	364221	623
624	SF 599	206939	Lode	MGI	08/26/2011	MGI	11/08/2011	364222	624
625	SF 600	207007	Lode	MGI	08/26/2011	MGI	11/09/2011	364223	625
626	SF 601	207008	Lode	MGI	08/26/2011	MGI	11/09/2011	364224	626
627	SF 602	207009	Lode	MGI	08/26/2011	MGI	11/09/2011	364225	627
628	SF 603	207010	Lode	MGI	08/26/2011	MGI	11/09/2011	364226	628
629	SF 604	207011	Lode	MGI	08/26/2011	MGI	11/09/2011	364227	629
630	SF 605	207012	Lode	MGI	08/26/2011	MGI	11/09/2011	364228	630
631	SF 606	207013	Lode	MGI	08/26/2011	MGI	11/09/2011	364229	631
632	SF 607	207014	Lode	MGI	08/26/2011	MGI	11/09/2011	364230	632
633	SF 608	207015	Lode	MGI	08/26/2011	MGI	11/09/2011	364231	633
634	SF 609	207016	Lode	MGI	08/26/2011	MGI	11/09/2011	364232	634
635	SF 610	207017	Lode	MGI	08/26/2011	MGI	11/09/2011	364233	635
636	SF 611	207018	Lode	MGI	08/26/2011	MGI	11/09/2011	364234	636
637	SF 612	207019	Lode	MGI	08/26/2011	MGI	11/09/2011	364235	637
638	SF 613	207020	Lode	MGI	08/25/2011	MGI	11/09/2011	364236	638
639	SF 614	207021	Lode	MGI	08/25/2011	MGI	11/09/2011	364237	639
640	SF 615	207022	Lode	MGI	08/25/2011	MGI	11/09/2011	364238	640
641	SF 616	207023	Lode	MGI	08/26/2011	MGI	11/09/2011	364239	641
642	SF 617	207024	Lode	MGI	08/26/2011	MGI	11/09/2011	364240	642
643	SF 618	207025	Lode	MGI	08/26/2011	MGI	11/09/2011	364241	643

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
644	SF 619	207026	Lode	MGI	08/26/2011	MGI	11/09/2011	364242	644
645	SF 620	207027	Lode	MGI	08/26/2011	MGI	11/09/2011	364243	645
646	SF 621	207028	Lode	MGI	08/26/2011	MGI	11/09/2011	364245	646
647	SF 622	207029	Lode	MGI	08/26/2011	MGI	11/09/2011	364246	647
648	SF 623	207030	Lode	MGI	08/26/2011	MGI	11/09/2011	364247	648
649	SF 624	207031	Lode	MGI	08/26/2011	MGI	11/09/2011	364248	649
650	SF 625	207032	Lode	MGI	08/26/2011	MGI	11/09/2011	364249	650
651	SF 626	207033	Lode	MGI	08/26/2011	MGI	11/09/2011	364250	651
652	SF 627	207034	Lode	MGI	08/26/2011	MGI	11/09/2011	364251	652
653	SF 628	207035	Lode	MGI	08/26/2011	MGI	11/09/2011	364252	653
654	SF 629	207036	Lode	MGI	08/26/2011	MGI	11/09/2011	364253	654
655	SF 630	207037	Lode	MGI	08/26/2011	MGI	11/09/2011	364254	655
656	SF 631	207038	Lode	MGI	08/26/2011	MGI	11/09/2011	364255	656
657	SF 632	207039	Lode	MGI	08/26/2011	MGI	11/09/2011	364256	657
658	SF 633	207040	Lode	MGI	08/26/2011	MGI	11/09/2011	364257	658
659	SF 634	207041	Lode	MGI	08/26/2011	MGI	11/09/2011	364258	659
660	SF 635	207042	Lode	MGI	08/26/2011	MGI	11/09/2011	364259	660
661	SF 636	207043	Lode	MGI	08/25/2011	MGI	11/09/2011	364260	661
662	SF 637	207044	Lode	MGI	08/25/2011	MGI	11/09/2011	364261	662
663	SF 638	207045	Lode	MGI	08/25/2011	MGI	11/09/2011	364262	663
664	SF 641	207046	Lode	MGI	08/27/2011	MGI	11/09/2011	364263	664
665	SF 642	207047	Lode	MGI	08/27/2011	MGI	11/09/2011	364264	665
666	SF 643	207048	Lode	MGI	08/27/2011	MGI	11/09/2011	364265	666
667	SF 644	207049	Lode	MGI	08/27/2011	MGI	11/09/2011	364266	667
668	SF 645	207050	Lode	MGI	08/25/2011	MGI	11/09/2011	364267	668
669	SF 646	207051	Lode	MGI	08/25/2011	MGI	11/09/2011	364268	669
670	SF 647	207052	Lode	MGI	08/25/2011	MGI	11/09/2011	364269	670
671	SF 648	207053	Lode	MGI	08/25/2011	MGI	11/09/2011	364270	671

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
672	SF 649	207054	Lode	MGI	08/25/2011	MGI	11/09/2011	364271	672
673	SF 650	207062	Lode	MGI	08/25/2011	MGI	11/09/2011	364272	673
674	SF 651	207063	Lode	MGI	08/25/2011	MGI	11/09/2011	364273	674
675	SF 652	207064	Lode	MGI	08/25/2011	MGI	11/09/2011	364274	675
676	SF 653	207065	Lode	MGI	08/25/2011	MGI	11/09/2011	364275	676
677	SF 654	207066	Lode	MGI	08/25/2011	MGI	11/09/2011	364276	677
678	SF 655	207067	Lode	MGI	08/25/2011	MGI	11/09/2011	364277	678
679	SF 656	207068	Lode	MGI	08/25/2011	MGI	11/09/2011	364278	679
680	SF 657	207069	Lode	MGI	08/25/2011	MGI	11/09/2011	364279	680
681	SF 658	207070	Lode	MGI	08/25/2011	MGI	11/09/2011	364280	681
682	SF 659	207071	Lode	MGI	08/25/2011	MGI	11/09/2011	364281	682
683	SF 660	207072	Lode	MGI	08/25/2011	MGI	11/09/2011	364282	683
684	SF 661	207073	Lode	MGI	08/25/2011	MGI	11/09/2011	364283	684
685	SF 662	207074	Lode	MGI	08/25/2011	MGI	11/09/2011	364284	685
686	SF 663	207075	Lode	MGI	08/25/2011	MGI	11/09/2011	364285	686
687	SF 664	207076	Lode	MGI	08/25/2011	MGI	11/09/2011	364286	687
688	SF 665	207077	Lode	MGI	08/25/2011	MGI	11/09/2011	364287	688
689	SF 666	207078	Lode	MGI	08/25/2011	MGI	11/09/2011	364288	689
690	SF 667	207079	Lode	MGI	08/25/2011	MGI	11/09/2011	364289	690
691	SF 668	207080	Lode	MGI	08/25/2011	MGI	11/09/2011	364290	691
692	SF 669	207081	Lode	MGI	08/25/2011	MGI	11/09/2011	364291	692
693	SF 670	207082	Lode	MGI	08/23/2011	MGI	11/09/2011	364292	693
694	SF 671	207083	Lode	MGI	08/23/2011	MGI	11/09/2011	364293	694
695	SF 672	207084	Lode	MGI	08/23/2011	MGI	11/09/2011	364294	695
696	SF 673	207085	Lode	MGI	08/23/2011	MGI	11/09/2011	364295	696
697	SF 674	207086	Lode	MGI	08/23/2011	MGI	11/09/2011	364296	697
698	SF 675	207087	Lode	MGI	08/18/2011	MGI	11/09/2011	364297	698
699	SF 676	207088	Lode	MGI	08/18/2011	MGI	11/09/2011	364298	699

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
700	SF 677	207089	Lode	MGI	08/23/2011	MGI	11/09/2011	364299	700
701	SF 678	207090	Lode	MGI	08/23/2011	MGI	11/09/2011	364300	701
702	SF 679	207091	Lode	MGI	08/23/2011	MGI	11/09/2011	364301	702
703	SF 680	207092	Lode	MGI	08/23/2011	MGI	11/09/2011	364302	703
704	SF 681	207093	Lode	MGI	08/23/2011	MGI	11/09/2011	364303	704
705	SF 682	207094	Lode	MGI	08/23/2011	MGI	11/09/2011	364304	705
706	SF 683	207095	Lode	MGI	08/23/2011	MGI	11/09/2011	364305	706
707	SF 684	207096	Lode	MGI	08/23/2011	MGI	11/09/2011	364306	707
708	SF 685	207097	Lode	MGI	08/23/2011	MGI	11/09/2011	364307	708
709	SF 686	207098	Lode	MGI	08/23/2011	MGI	11/09/2011	364308	709
710	SF 687	207099	Lode	MGI	08/23/2011	MGI	11/09/2011	364309	710
711	SF 688	207100	Lode	MGI	08/23/2011	MGI	11/09/2011	364310	711
712	SF 689	207101	Lode	MGI	08/23/2011	MGI	11/09/2011	364311	712
713	SF 690	207102	Lode	MGI	08/23/2011	MGI	11/09/2011	364312	713
714	SF 691	207103	Lode	MGI	08/23/2011	MGI	11/09/2011	364313	714
715	SF 692	207104	Lode	MGI	08/23/2011	MGI	11/09/2011	364314	715
716	SF 693	207105	Lode	MGI	08/23/2011	MGI	11/09/2011	364315	716
717	SF 694	207106	Lode	MGI	08/23/2011	MGI	11/09/2011	364316	717
718	SF 695	207107	Lode	MGI	08/23/2011	MGI	11/09/2011	364317	718
719	SF 696	207108	Lode	MGI	08/23/2011	MGI	11/09/2011	364318	719
720	SF 697	207109	Lode	MGI	08/23/2011	MGI	11/09/2011	364319	720
721	SF 698	207110	Lode	MGI	08/23/2011	MGI	11/09/2011	364320	721
722	SF 699	207111	Lode	MGI	08/23/2011	MGI	11/09/2011	364321	722
723	SF 700	207112	Lode	MGI	08/23/2011	MGI	11/09/2011	364322	723
724	SF 701	207113	Lode	MGI	08/23/2011	MGI	11/09/2011	364329	724
725	SF 702	207114	Lode	MGI	08/23/2011	MGI	11/09/2011	364330	725
726	SF 703	207115	Lode	MGI	08/23/2011	MGI	11/09/2011	364331	726
727	SF 704	207174	Lode	MGI	09/19/2011	MGI	11/10/2011	364479	727

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
728	SF 705	207175	Lode	MGI	09/19/2011	MGI	11/10/2011	364480	728
729	SF 706	207176	Lode	MGI	09/19/2011	MGI	11/10/2011	364481	729
730	SF 707	207177	Lode	MGI	09/19/2011	MGI	11/10/2011	364482	730
731	SF 708	207178	Lode	MGI	09/19/2011	MGI	11/10/2011	364483	731
732	SF 709	207179	Lode	MGI	09/19/2011	MGI	11/10/2011	364484	732
733	SF 710	207180	Lode	MGI	09/19/2011	MGI	11/10/2011	364485	733
734	SF 711	207181	Lode	MGI	09/19/2011	MGI	11/10/2011	364486	734
735	SF 712	207182	Lode	MGI	09/19/2011	MGI	11/10/2011	364487	735
736	SF 713	207183	Lode	MGI	09/19/2011	MGI	11/10/2011	364488	736
737	SF 714	207184	Lode	MGI	09/19/2011	MGI	11/10/2011	364489	737
738	SF 715	207185	Lode	MGI	09/19/2011	MGI	11/10/2011	364490	738
739	SF 716	207186	Lode	MGI	09/19/2011	MGI	11/10/2011	364491	739
740	SF 717	207187	Lode	MGI	09/19/2011	MGI	11/10/2011	364492	740
741	SF 718	207188	Lode	MGI	09/19/2011	MGI	11/10/2011	364493	741
742	SF 719	207189	Lode	MGI	09/19/2011	MGI	11/10/2011	364494	742
743	SF 720	207190	Lode	MGI	09/19/2011	MGI	11/10/2011	364495	743
744	SF 721	206940	Lode	MGI	08/30/2011	MGI	11/09/2011	364332	744
745	SF 722	206941	Lode	MGI	08/30/2011	MGI	11/09/2011	364333	745
746	SF 723	206942	Lode	MGI	08/30/2011	MGI	11/09/2011	364334	746
747	SF 724	206943	Lode	MGI	08/30/2011	MGI	11/09/2011	364335	747
748	SF 725	206944	Lode	MGI	08/30/2011	MGI	11/09/2011	364336	748
749	SF 726	206945	Lode	MGI	08/30/2011	MGI	11/09/2011	364337	749
750	SF 727	206946	Lode	MGI	08/30/2011	MGI	11/09/2011	364338	750
751	SF 728	206947	Lode	MGI	08/30/2011	MGI	11/09/2011	364339	751
752	SF 729	206948	Lode	MGI	08/30/2011	MGI	11/09/2011	364340	752
753	SF 730	206949	Lode	MGI	08/30/2011	MGI	11/09/2011	364341	753
754	SF 731	206950	Lode	MGI	08/30/2011	MGI	11/09/2011	364342	754
755	SF 732	206951	Lode	MGI	08/30/2011	MGI	11/09/2011	364343	755

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
756	SF 733	206952	Lode	MGI	08/30/2011	MGI	11/09/2011	364344	756
757	SF 734	206953	Lode	MGI	08/30/2011	MGI	11/09/2011	364345	757
758	SF 735	206954	Lode	MGI	08/29/2011	MGI	11/09/2011	364346	758
759	SF 736	206955	Lode	MGI	08/29/2011	MGI	11/09/2011	364347	759
760	SF 737	206956	Lode	MGI	08/29/2011	MGI	11/09/2011	364348	760
761	SF 738	206957	Lode	MGI	08/29/2011	MGI	11/09/2011	364349	761
762	SF 739	206958	Lode	MGI	08/29/2011	MGI	11/09/2011	364350	762
763	SF 740	206959	Lode	MGI	08/29/2011	MGI	11/09/2011	364351	763
764	SF 741	206960	Lode	MGI	08/29/2011	MGI	11/09/2011	364352	764
765	SF 742	206961	Lode	MGI	08/29/2011	MGI	11/09/2011	364353	765
766	SF 743	206962	Lode	MGI	08/29/2011	MGI	11/09/2011	364354	766
767	SF 744	206963	Lode	MGI	08/29/2011	MGI	11/09/2011	364355	767
768	SF 745	206964	Lode	MGI	08/29/2011	MGI	11/09/2011	364356	768
769	SF 746	206965	Lode	MGI	08/29/2011	MGI	11/09/2011	364357	769
770	SF 747	206966	Lode	MGI	08/29/2011	MGI	11/09/2011	364358	770
771	SF 748	206967	Lode	MGI	08/29/2011	MGI	11/09/2011	364359	771
772	SF 749	206968	Lode	MGI	08/29/2011	MGI	11/09/2011	364360	772
773	SF 750	206969	Lode	MGI	08/29/2011	MGI	11/09/2011	364361	773
774	SF 751	206970	Lode	MGI	08/29/2011	MGI	11/09/2011	364362	774
775	SF 752	206971	Lode	MGI	08/29/2011	MGI	11/09/2011	364363	775
776	SF 753	206972	Lode	MGI	08/30/2011	MGI	11/09/2011	364364	776
777	SF 754	206973	Lode	MGI	08/30/2011	MGI	11/09/2011	364365	777
778	SF 755	206974	Lode	MGI	08/30/2011	MGI	11/09/2011	364366	778
779	SF 756	206975	Lode	MGI	08/30/2011	MGI	11/09/2011	364367	779
780	SF 757	206976	Lode	MGI	08/30/2011	MGI	11/09/2011	364368	780
781	SF 758	206977	Lode	MGI	08/30/2011	MGI	11/09/2011	364369	781
782	SF 759	206978	Lode	MGI	08/30/2011	MGI	11/09/2011	364370	782
783	SF 760	206979	Lode	MGI	08/30/2011	MGI	11/09/2011	364371	783

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
784	SF 761	206980	Lode	MGI	08/30/2011	MGI	11/09/2011	364372	784
785	SF 762	206981	Lode	MGI	08/30/2011	MGI	11/09/2011	364373	785
786	SF 763	206982	Lode	MGI	08/30/2011	MGI	11/09/2011	364374	786
787	SF 764	206983	Lode	MGI	08/30/2011	MGI	11/09/2011	364375	787
788	SF 765	206984	Lode	MGI	08/30/2011	MGI	11/09/2011	364376	788
789	SF 766	206985	Lode	MGI	08/30/2011	MGI	11/09/2011	364377	789
790	SF 767	206986	Lode	MGI	08/30/2011	MGI	11/09/2011	364378	790
791	SF 768	206987	Lode	MGI	08/30/2011	MGI	11/09/2011	364379	791
792	SF 769	206988	Lode	MGI	08/30/2011	MGI	11/09/2011	364380	792
793	SF 770	206989	Lode	MGI	08/30/2011	MGI	11/09/2011	364381	793
794	SF 771	206990	Lode	MGI	08/30/2011	MGI	11/09/2011	364382	794
795	SF 772	206991	Lode	MGI	08/30/2011	MGI	11/09/2011	364383	795
796	SF 773	206992	Lode	MGI	08/30/2011	MGI	11/09/2011	364384	796
797	SF 774	206993	Lode	MGI	08/30/2011	MGI	11/09/2011	364385	797
798	SF 775	206994	Lode	MGI	08/30/2011	MGI	11/09/2011	364386	798
799	SF 776	206995	Lode	MGI	08/30/2011	MGI	11/09/2011	364387	799
800	SF 777	206996	Lode	MGI	08/29/2011	MGI	11/09/2011	364388	800
801	SF 778	206997	Lode	MGI	08/29/2011	MGI	11/09/2011	364389	801
802	SF 779	206998	Lode	MGI	08/29/2011	MGI	11/09/2011	364390	802
803	SF 780	206999	Lode	MGI	08/29/2011	MGI	11/09/2011	364391	803
804	SF 781	207000	Lode	MGI	08/29/2011	MGI	11/09/2011	364392	804
805	SF 782	207001	Lode	MGI	08/29/2011	MGI	11/09/2011	364393	805
806	SF 783	207002	Lode	MGI	08/29/2011	MGI	11/09/2011	364394	806
807	SF 784	207003	Lode	MGI	08/29/2011	MGI	11/09/2011	364395	807
808	SF 785	207004	Lode	MGI	08/29/2011	MGI	11/09/2011	364396	808
809	SF 786	207005	Lode	MGI	08/30/2011	MGI	11/09/2011	364397	809
810	SF 787	207006	Lode	MGI	08/30/2011	MGI	11/09/2011	364398	810
811	SF 788	207191	Lode	MGI	09/07/2011	MGI	11/10/2011	364496	811

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
812	SF 789	207192	Lode	MGI	09/07/2011	MGI	11/10/2011	364497	812
813	SF 790	207193	Lode	MGI	09/07/2011	MGI	11/10/2011	364498	813
814	SF 791	207194	Lode	MGI	09/07/2011	MGI	11/10/2011	364499	814
815	SF 792	207195	Lode	MGI	09/07/2011	MGI	11/10/2011	364500	815
816	SF 793	207196	Lode	MGI	09/07/2011	MGI	11/10/2011	364501	816
817	SF 794	207197	Lode	MGI	09/07/2011	MGI	11/10/2011	364502	817
818	SF 795	207198	Lode	MGI	09/07/2011	MGI	11/10/2011	364503	818
819	SF 796	207199	Lode	MGI	09/07/2011	MGI	11/10/2011	364504	819
820	SF 797	207200	Lode	MGI	09/07/2011	MGI	11/10/2011	364505	820
821	SF 798	207201	Lode	MGI	09/07/2011	MGI	11/10/2011	364506	821
822	SF 799	207202	Lode	MGI	09/07/2011	MGI	11/10/2011	364507	822
823	SF 800	207203	Lode	MGI	09/07/2011	MGI	11/15/2011	364581	823
824	SF 801	207204	Lode	MGI	09/07/2011	MGI	11/15/2011	364582	824
825	SF 802	207205	Lode	MGI	09/07/2011	MGI	11/15/2011	364583	825
826	SF 803	207055	Lode	MGI	08/31/2011	MGI	11/15/2011	364408	826
827	SF 804	207056	Lode	MGI	08/31/2011	MGI	11/15/2011	364409	827
828	SF 805	207057	Lode	MGI	08/31/2011	MGI	11/15/2011	364410	828
829	SF 806	207058	Lode	MGI	08/31/2011	MGI	11/15/2011	364411	829
830	SF 807	207059	Lode	MGI	08/31/2011	MGI	11/15/2011	364412	830
831	SF 808	207060	Lode	MGI	08/31/2011	MGI	11/15/2011	364413	831
832	SF 809	207061	Lode	MGI	08/31/2011	MGI	11/15/2011	364414	832
833	SF 810	207206	Lode	MGI	09/07/2011	MGI	11/15/2011	364584	833
834	SF 811	207207	Lode	MGI	09/07/2011	MGI	11/15/2011	364585	834
835	SF 812	207208	Lode	MGI	09/07/2011	MGI	11/15/2011	364586	835
836	SF 813	207209	Lode	MGI	09/07/2011	MGI	11/15/2011	364587	836
837	SF 814	207210	Lode	MGI	09/07/2011	MGI	11/15/2011	364588	837
838	SF 815	207211	Lode	MGI	09/07/2011	MGI	11/15/2011	364589	838
839	SF 816	207212	Lode	MGI	09/07/2011	MGI	11/15/2011	364590	839

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
840	SF 817	207213	Lode	MGI	09/07/2011	MGI	11/15/2011	364591	840
841	SF 818	207214	Lode	MGI	09/07/2011	MGI	11/15/2011	364592	841
842	SF 819	207215	Lode	MGI	09/07/2011	MGI	11/15/2011	364593	842
843	SF 820	207216	Lode	MGI	09/07/2011	MGI	11/15/2011	364594	843
844	SF 821	207217	Lode	MGI	09/07/2011	MGI	11/15/2011	364595	844
845	SF 822	207218	Lode	MGI	09/07/2011	MGI	11/15/2011	364596	845
846	SF 823	207219	Lode	MGI	09/07/2011	MGI	11/15/2011	364597	846
847	SF 824	207220	Lode	MGI	09/07/2011	MGI	11/15/2011	364598	847
848	SF 825	207221	Lode	MGI	09/07/2011	MGI	11/15/2011	364599	848
849	SF 826	207116	Lode	MGI	08/31/2011	MGI	11/15/2011	364415	849
850	SF 827	207117	Lode	MGI	08/31/2011	MGI	11/15/2011	364416	850
851	SF 828	207118	Lode	MGI	08/31/2011	MGI	11/15/2011	364417	851
852	SF 829	207119	Lode	MGI	08/31/2011	MGI	11/15/2011	364418	852
853	SF 830	207120	Lode	MGI	08/31/2011	MGI	11/15/2011	364419	853
854	SF 831	207121	Lode	MGI	08/31/2011	MGI	11/15/2011	364420	854
855	SF 832	207122	Lode	MGI	08/31/2011	MGI	11/15/2011	364421	855
856	SF 833	207123	Lode	MGI	08/31/2011	MGI	11/15/2011	364422	856
857	SF 834	207222	Lode	MGI	09/07/2011	MGI	11/15/2011	364600	857
858	SF 835	207223	Lode	MGI	09/07/2011	MGI	11/15/2011	364601	858
859	SF 836	207224	Lode	MGI	09/07/2011	MGI	11/15/2011	364602	859
860	SF 837	207225	Lode	MGI	09/07/2011	MGI	11/15/2011	364603	860
861	SF 838	207226	Lode	MGI	09/07/2011	MGI	11/15/2011	364604	861
862	SF 839	207227	Lode	MGI	09/07/2011	MGI	11/15/2011	364605	862
863	SF 840	207228	Lode	MGI	09/07/2011	MGI	11/15/2011	364606	863
864	SF 841	207229	Lode	MGI	09/07/2011	MGI	11/15/2011	364607	864
865	SF 842	207230	Lode	MGI	09/07/2011	MGI	11/15/2011	364608	865
866	SF 843	207231	Lode	MGI	09/07/2011	MGI	11/15/2011	364609	866
867	SF 844	207232	Lode	MGI	09/07/2011	MGI	11/15/2011	364610	867

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
868	SF 845	207233	Lode	MGI	09/07/2011	MGI	11/15/2011	364611	868
869	SF 846	207234	Lode	MGI	09/07/2011	MGI	11/15/2011	364612	869
870	SF 847	207235	Lode	MGI	09/07/2011	MGI	11/15/2011	364613	870
871	SF 848	207124	Lode	MGI	08/31/2011	MGI	11/15/2011	364423	871
872	SF 849	207125	Lode	MGI	08/31/2011	MGI	11/15/2011	364424	872
873	SF 850	207126	Lode	MGI	08/31/2011	MGI	11/15/2011	364425	873
874	SF 851	207127	Lode	MGI	08/31/2011	MGI	11/15/2011	364426	874
875	SF 852	207128	Lode	MGI	08/31/2011	MGI	11/15/2011	364427	875
876	SF 853	207129	Lode	MGI	08/31/2011	MGI	11/15/2011	364428	876
877	SF 854	207130	Lode	MGI	08/31/2011	MGI	11/15/2011	364429	877
878	SF 855	207131	Lode	MGI	08/31/2011	MGI	11/15/2011	364430	878
879	SF 856	207236	Lode	MGI	09/07/2011	MGI	11/15/2011	364614	879
880	SF 857	207237	Lode	MGI	09/07/2011	MGI	11/15/2011	364615	880
881	SF 858	207238	Lode	MGI	09/07/2011	MGI	11/15/2011	364616	881
882	SF 859	207239	Lode	MGI	09/07/2011	MGI	11/15/2011	364617	882
883	SF 860	207240	Lode	MGI	09/07/2011	MGI	11/15/2011	364618	883
884	SF 861	207241	Lode	MGI	09/07/2011	MGI	11/15/2011	364619	884
885	SF 862	207242	Lode	MGI	09/07/2011	MGI	11/15/2011	364620	885
886	SF 863	207243	Lode	MGI	09/07/2011	MGI	11/15/2011	364621	886
887	SF 864	207244	Lode	MGI	09/07/2011	MGI	11/15/2011	364622	887
888	SF 865	207245	Lode	MGI	09/07/2011	MGI	11/15/2011	364623	888
889	SF 866	207246	Lode	MGI	09/07/2011	MGI	11/15/2011	364624	889
890	SF 867	207247	Lode	MGI	09/07/2011	MGI	11/15/2011	364625	890
891	SF 868	207132	Lode	MGI	08/31/2011	MGI	11/15/2011	364431	891
892	SF 869	207133	Lode	MGI	08/31/2011	MGI	11/15/2011	364432	892
893	SF 870	207134	Lode	MGI	08/31/2011	MGI	11/15/2011	364433	893
894	SF 871	207135	Lode	MGI	08/31/2011	MGI	11/15/2011	364434	894
895	SF 872	207136	Lode	MGI	08/31/2011	MGI	11/15/2011	364435	895

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
896	SF 873	207137	Lode	MGI	08/31/2011	MGI	11/15/2011	364436	896
897	SF 874	207138	Lode	MGI	08/31/2011	MGI	11/15/2011	364437	897
898	SF 875	207139	Lode	MGI	08/31/2011	MGI	11/15/2011	364438	898
899	SF 876	207248	Lode	MGI	09/08/2011	MGI	11/15/2011	364628	899
900	SF 877	207249	Lode	MGI	09/08/2011	MGI	11/15/2011	364629	900
901	SF 878	207250	Lode	MGI	09/08/2011	MGI	11/15/2011	364630	901
902	SF 879	207251	Lode	MGI	09/08/2011	MGI	11/15/2011	364631	902
903	SF 880	207252	Lode	MGI	09/08/2011	MGI	11/15/2011	364632	903
904	SF 881	207253	Lode	MGI	09/08/2011	MGI	11/15/2011	364633	904
905	SF 882	207254	Lode	MGI	09/08/2011	MGI	11/15/2011	364634	905
906	SF 883	207255	Lode	MGI	09/08/2011	MGI	11/15/2011	364635	906
907	SF 884	207256	Lode	MGI	09/08/2011	MGI	11/15/2011	364636	907
908	SF 885	207257	Lode	MGI	09/08/2011	MGI	11/15/2011	364637	908
909	SF 886	207258	Lode	MGI	09/08/2011	MGI	11/15/2011	364638	909
910	SF 887	207259	Lode	MGI	09/08/2011	MGI	11/15/2011	364639	910
911	SF 888	207260	Lode	MGI	09/08/2011	MGI	11/15/2011	364640	911
912	SF 889	207261	Lode	MGI	09/08/2011	MGI	11/15/2011	364641	912
913	SF 890	207262	Lode	MGI	09/08/2011	MGI	11/15/2011	364642	913
914	SF 891	207263	Lode	MGI	09/08/2011	MGI	11/15/2011	364643	914
915	SF 892	207264	Lode	MGI	09/08/2011	MGI	11/15/2011	364644	915
916	SF 893	207265	Lode	MGI	09/08/2011	MGI	11/15/2011	364645	916
917	SF 894	207266	Lode	MGI	09/08/2011	MGI	11/15/2011	364646	917
918	SF 895	207267	Lode	MGI	09/08/2011	MGI	11/15/2011	364647	918
919	SF 896	207268	Lode	MGI	09/08/2011	MGI	11/15/2011	364648	919
920	SF 897	207269	Lode	MGI	09/08/2011	MGI	11/15/2011	364649	920
921	SF 898	207270	Lode	MGI	09/08/2011	MGI	11/15/2011	364650	921
922	SF 899	207271	Lode	MGI	09/08/2011	MGI	11/15/2011	364651	922
923	SF 900	207272	Lode	MGI	09/08/2011	MGI	11/17/2011	364658	923

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
924	SF 901	207273	Lode	MGI	09/08/2011	MGI	11/17/2011	364692	924
925	SF 902	207274	Lode	MGI	09/08/2011	MGI	11/17/2011	364695	925
926	SF 903	207275	Lode	MGI	09/08/2011	MGI	11/17/2011	364696	926
927	SF 904	207276	Lode	MGI	09/08/2011	MGI	11/17/2011	364697	927
928	SF 905	207277	Lode	MGI	09/08/2011	MGI	11/17/2011	364698	928
929	SF 906	207278	Lode	MGI	09/08/2011	MGI	11/17/2011	364699	929
930	SF 907	207279	Lode	MGI	09/08/2011	MGI	11/17/2011	364700	930
931	SF 908	207280	Lode	MGI	09/08/2011	MGI	11/17/2011	364701	931
932	SF 909	207281	Lode	MGI	09/08/2011	MGI	11/17/2011	364702	932
933	SF 910	207282	Lode	MGI	09/08/2011	MGI	11/17/2011	364703	933
934	SF 911	207283	Lode	MGI	09/08/2011	MGI	11/17/2011	364704	934
935	SF 912	207284	Lode	MGI	09/08/2011	MGI	11/17/2011	364705	935
936	SF 913	207285	Lode	MGI	09/08/2011	MGI	11/17/2011	364706	936
937	SF 914	207286	Lode	MGI	09/08/2011	MGI	11/17/2011	364707	937
938	SF 915	207287	Lode	MGI	09/08/2011	MGI	11/17/2011	364708	938
939	SF 916	207288	Lode	MGI	09/08/2011	MGI	11/17/2011	364709	939
940	SF 917	207289	Lode	MGI	09/08/2011	MGI	11/17/2011	364710	940
941	SF 918	207290	Lode	MGI	09/08/2011	MGI	11/17/2011	364711	941
942	SF 919	207291	Lode	MGI	09/08/2011	MGI	11/17/2011	364712	942
943	SF 920	207140	Lode	MGI	08/27/2011	MGI	11/17/2011	364439	943
944	SF 921	207141	Lode	MGI	08/30/2011	MGI	11/17/2011	364440	944
945	SF 922	207292	Lode	MGI	09/08/2011	MGI	11/17/2011	364713	945
946	SF 923	207293	Lode	MGI	09/08/2011	MGI	11/17/2011	364714	946
947	SF 924	207294	Lode	MGI	09/08/2011	MGI	11/17/2011	364715	947
948	SF 925	207295	Lode	MGI	09/08/2011	MGI	11/17/2011	364716	948
949	SF 926	207296	Lode	MGI	09/08/2011	MGI	11/17/2011	364717	949
950	SF 927	207297	Lode	MGI	09/08/2011	MGI	11/17/2011	364718	950
951	SF 928	207298	Lode	MGI	09/08/2011	MGI	11/17/2011	364719	951

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
952	SF 929	207299	Lode	MGI	09/08/2011	MGI	11/17/2011	364720	952
953	SF 930	207300	Lode	MGI	09/08/2011	MGI	11/17/2011	364721	953
954	SF 931	207301	Lode	MGI	09/08/2011	MGI	11/17/2011	364722	954
955	SF 932	207302	Lode	MGI	09/08/2011	MGI	11/17/2011	364723	955
956	SF 933	207303	Lode	MGI	09/08/2011	MGI	11/17/2011	364724	956
957	SF 934	207304	Lode	MGI	09/08/2011	MGI	11/17/2011	364725	957
958	SF 935	207305	Lode	MGI	09/08/2011	MGI	11/17/2011	364726	958
959	SF 936	207306	Lode	MGI	09/08/2011	MGI	11/17/2011	364727	959
960	SF 937	207307	Lode	MGI	09/08/2011	MGI	11/17/2011	364728	960
961	SF 938	207308	Lode	MGI	09/08/2011	MGI	11/17/2011	364729	961
962	SF 939	207309	Lode	MGI	09/08/2011	MGI	11/17/2011	364730	962
963	SF 940	207310	Lode	MGI	09/08/2011	MGI	11/17/2011	364731	963
964	SF 941	207311	Lode	MGI	09/09/2011	MGI	11/17/2011	364732	964
965	SF 942	207312	Lode	MGI	09/09/2011	MGI	11/17/2011	364733	965
966	SF 943	207313	Lode	MGI	09/09/2011	MGI	11/17/2011	364734	966
967	SF 944	207314	Lode	MGI	09/09/2011	MGI	11/17/2011	364735	967
968	SF 945	207315	Lode	MGI	09/09/2011	MGI	11/17/2011	364736	968
969	SF 946	207316	Lode	MGI	09/09/2011	MGI	11/17/2011	364737	969
970	SF 947	207317	Lode	MGI	09/09/2011	MGI	11/17/2011	364738	970
971	SF 948	207318	Lode	MGI	09/09/2011	MGI	11/17/2011	364739	971
972	SF 949	207319	Lode	MGI	09/09/2011	MGI	11/17/2011	364740	972
973	SF 950	207320	Lode	MGI	09/09/2011	MGI	11/17/2011	364741	973
974	SF 951	207321	Lode	MGI	09/09/2011	MGI	11/17/2011	364742	974
975	SF 952	207322	Lode	MGI	09/09/2011	MGI	11/17/2011	364743	975
976	SF 953	207323	Lode	MGI	09/09/2011	MGI	11/17/2011	364744	976
977	SF 954	207324	Lode	MGI	09/08/2011	MGI	11/17/2011	364745	977
978	SF 955	207325	Lode	MGI	09/08/2011	MGI	11/17/2011	364746	978
979	SF 956	207326	Lode	MGI	09/08/2011	MGI	11/17/2011	364747	979

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
980	SF 957	207327	Lode	MGI	09/09/2011	MGI	11/17/2011	364748	980
981	SF 958	207328	Lode	MGI	09/09/2011	MGI	11/17/2011	364749	981
982	SF 959	207329	Lode	MGI	09/09/2011	MGI	11/17/2011	364750	982
983	SF 960	207330	Lode	MGI	09/09/2011	MGI	11/17/2011	364751	983
984	SF 961	207331	Lode	MGI	09/09/2011	MGI	11/17/2011	364752	984
985	SF 962	207332	Lode	MGI	09/09/2011	MGI	11/17/2011	364753	985
986	SF 963	207333	Lode	MGI	09/09/2011	MGI	11/17/2011	364754	986
987	SF 964	207334	Lode	MGI	09/09/2011	MGI	11/17/2011	364755	987
988	SF 965	207335	Lode	MGI	09/09/2011	MGI	11/17/2011	364756	988
989	SF 966	207336	Lode	MGI	09/09/2011	MGI	11/17/2011	364757	989
990	SF 967	207337	Lode	MGI	09/09/2011	MGI	11/17/2011	364758	990
991	SF 968	207338	Lode	MGI	09/09/2011	MGI	11/17/2011	364769	991
992	SF 969	207339	Lode	MGI	09/09/2011	MGI	11/17/2011	364770	992
993	SF 970	207340	Lode	MGI	09/09/2011	MGI	11/17/2011	364771	993
994	SF 971	207341	Lode	MGI	09/09/2011	MGI	11/17/2011	364772	994
995	SF 972	207342	Lode	MGI	09/09/2011	MGI	11/17/2011	364773	995
996	SF 973	207343	Lode	MGI	09/09/2011	MGI	11/17/2011	364774	996
997	SF 974	207344	Lode	MGI	09/09/2011	MGI	11/17/2011	364775	997
998	SF 975	207345	Lode	MGI	09/09/2011	MGI	11/17/2011	364776	998
999	SF 976	207346	Lode	MGI	09/09/2011	MGI	11/17/2011	364777	999
1000	SF 977	207347	Lode	MGI	09/09/2011	MGI	11/17/2011	364778	1000
1001	SF 978	207348	Lode	MGI	09/09/2011	MGI	11/17/2011	364779	1001
1002	SF 979	207349	Lode	MGI	09/09/2011	MGI	11/17/2011	364780	1002
1003	SF 980	207350	Lode	MGI	09/09/2011	MGI	11/17/2011	364781	1003
1004	SF 981	207351	Lode	MGI	09/09/2011	MGI	11/17/2011	364782	1004
1005	SF 982	207352	Lode	MGI	09/09/2011	MGI	11/17/2011	364783	1005
1006	SF 983	207353	Lode	MGI	09/09/2011	MGI	11/17/2011	364784	1006
1007	SF 984	207354	Lode	MGI	09/09/2011	MGI	11/17/2011	364785	1007

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1008	SF 985	207355	Lode	MGI	09/09/2011	MGI	11/17/2011	364786	1008
1009	SF 986	207356	Lode	MGI	09/09/2011	MGI	11/17/2011	364787	1009
1010	SF 987	207357	Lode	MGI	09/09/2011	MGI	11/17/2011	364788	1010
1011	SF 988	207358	Lode	MGI	09/09/2011	MGI	11/17/2011	364789	1011
1012	SF 989	207359	Lode	MGI	09/09/2011	MGI	11/17/2011	364790	1012
1013	SF 990	207360	Lode	MGI	09/09/2011	MGI	11/17/2011	364791	1013
1014	SF 991	207361	Lode	MGI	09/09/2011	MGI	11/17/2011	364792	1014
1015	SF 992	207362	Lode	MGI	09/09/2011	MGI	11/17/2011	364793	1015
1016	SF 993	207363	Lode	MGI	09/10/2011	MGI	11/17/2011	364794	1016
1017	SF 994	207364	Lode	MGI	09/10/2011	MGI	11/17/2011	364795	1017
1018	SF 995	207365	Lode	MGI	09/10/2011	MGI	11/17/2011	364801	1018
1019	SF 996	207366	Lode	MGI	09/10/2011	MGI	11/17/2011	364802	1019
1020	SF 997	207367	Lode	MGI	09/10/2011	MGI	11/17/2011	364803	1020
1021	SF 998	207368	Lode	MGI	09/10/2011	MGI	11/17/2011	364804	1021
1022	SF 999	207369	Lode	MGI	09/10/2011	MGI	11/17/2011	364805	1022
1023	SF 1000	207370	Lode	MGI	09/10/2011	MGI	11/18/2011	364820	1023
1024	SF 1001	207371	Lode	MGI	09/10/2011	MGI	11/18/2011	364821	1024
1025	SF 1002	207372	Lode	MGI	09/10/2011	MGI	11/18/2011	364822	1025
1026	SF 1003	207373	Lode	MGI	09/10/2011	MGI	11/18/2011	364823	1026
1027	SF 1004	207374	Lode	MGI	09/10/2011	MGI	11/18/2011	364824	1027
1028	SF 1005	207375	Lode	MGI	09/10/2011	MGI	11/18/2011	364825	1028
1029	SF 1006	207376	Lode	MGI	09/10/2011	MGI	11/18/2011	364826	1029
1030	SF 1007	207377	Lode	MGI	09/10/2011	MGI	11/18/2011	364832	1030
1031	SF 1008	207378	Lode	MGI	09/10/2011	MGI	11/18/2011	364833	1031
1032	SF 1009	207379	Lode	MGI	09/10/2011	MGI	11/18/2011	364834	1032
1033	SF 1010	207380	Lode	MGI	09/10/2011	MGI	11/18/2011	364835	1033
1034	SF 1011	207381	Lode	MGI	09/10/2011	MGI	11/18/2011	364836	1034
1035	SF 1012	207382	Lode	MGI	09/10/2011	MGI	11/18/2011	364837	1035

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1036	SF 1013	207383	Lode	MGI	09/10/2011	MGI	11/18/2011	364838	1036
1037	SF 1014	207384	Lode	MGI	09/10/2011	MGI	11/18/2011	364839	1037
1038	SF 1015	207385	Lode	MGI	09/10/2011	MGI	11/18/2011	364840	1038
1039	SF 1016	207386	Lode	MGI	09/10/2011	MGI	11/18/2011	364841	1039
1040	SF 1017	207387	Lode	MGI	09/10/2011	MGI	11/18/2011	364842	1040
1041	SF 1018	207388	Lode	MGI	09/10/2011	MGI	11/18/2011	364843	1041
1042	SF 1019	207389	Lode	MGI	09/10/2011	MGI	11/18/2011	364844	1042
1043	SF 1020	207390	Lode	MGI	09/10/2011	MGI	11/18/2011	364845	1043
1044	SF 1021	207391	Lode	MGI	09/10/2011	MGI	11/18/2011	364846	1044
1045	SF 1022	207392	Lode	MGI	09/10/2011	MGI	11/18/2011	364847	1045
1046	SF 1023	207393	Lode	MGI	09/10/2011	MGI	11/18/2011	364848	1046
1047	SF 1024	207394	Lode	MGI	09/10/2011	MGI	11/18/2011	364849	1047
1048	SF 1025	207395	Lode	MGI	09/10/2011	MGI	11/18/2011	364850	1048
1049	SF 1026	207396	Lode	MGI	09/10/2011	MGI	11/18/2011	364851	1049
1050	SF 1027	207397	Lode	MGI	09/10/2011	MGI	11/18/2011	364852	1050
1051	SF 1028	207398	Lode	MGI	09/12/2011	MGI	11/18/2011	364853	1051
1052	SF 1029	207399	Lode	MGI	09/12/2011	MGI	11/18/2011	364854	1052
1053	SF 1030	207400	Lode	MGI	09/12/2011	MGI	11/18/2011	364855	1053
1054	SF 1031	207401	Lode	MGI	09/12/2011	MGI	11/18/2011	364856	1054
1055	SF 1032	207402	Lode	MGI	09/12/2011	MGI	11/18/2011	364857	1055
1056	SF 1033	207403	Lode	MGI	09/12/2011	MGI	11/18/2011	364858	1056
1057	SF 1034	207404	Lode	MGI	09/12/2011	MGI	11/18/2011	364859	1057
1058	SF 1035	207405	Lode	MGI	09/12/2011	MGI	11/18/2011	364860	1058
1059	SF 1036	207406	Lode	MGI	09/17/2011	MGI	11/18/2011	364861	1059
1060	SF 1037	207407	Lode	MGI	09/17/2011	MGI	11/18/2011	364862	1060
1061	SF 1038	207408	Lode	MGI	09/17/2011	MGI	11/18/2011	364863	1061
1062	SF 1039	207409	Lode	MGI	09/17/2011	MGI	11/18/2011	364864	1062
1063	SF 1040	207410	Lode	MGI	09/17/2011	MGI	11/18/2011	364865	1063

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1064	SF 1041	207411	Lode	MGI	09/08/2011	MGI	11/18/2011	364866	1064
1065	SF 1042	207412	Lode	MGI	09/12/2011	MGI	11/18/2011	364867	1065
1066	SF 1043	207413	Lode	MGI	09/12/2011	MGI	11/18/2011	364871	1066
1067	SF 1044	207414	Lode	MGI	09/12/2011	MGI	11/18/2011	364872	1067
1068	SF 1045	207415	Lode	MGI	09/12/2011	MGI	11/18/2011	364873	1068
1069	SF 1046	207416	Lode	MGI	09/12/2011	MGI	11/18/2011	364874	1069
1070	SF 1047	207417	Lode	MGI	09/12/2011	MGI	11/18/2011	364875	1070
1071	SF 1048	207418	Lode	MGI	09/12/2011	MGI	11/18/2011	364876	1071
1072	SF 1049	207419	Lode	MGI	09/12/2011	MGI	11/18/2011	364877	1072
1073	SF 1050	207420	Lode	MGI	09/12/2011	MGI	11/18/2011	364878	1073
1074	SF 1051	207421	Lode	MGI	09/17/2011	MGI	11/18/2011	364879	1074
1075	SF 1052	207422	Lode	MGI	09/17/2011	MGI	11/18/2011	364880	1075
1076	SF 1053	207423	Lode	MGI	09/17/2011	MGI	11/18/2011	364881	1076
1077	SF 1054	207424	Lode	MGI	09/17/2011	MGI	11/18/2011	364882	1077
1078	SF 1055	207425	Lode	MGI	09/17/2011	MGI	11/18/2011	364883	1078
1079	SF 1056	207426	Lode	MGI	09/17/2011	MGI	11/18/2011	364884	1079
1080	SF 1057	207427	Lode	MGI	09/17/2011	MGI	11/18/2011	364885	1080
1081	SF 1058	207428	Lode	MGI	09/17/2011	MGI	11/18/2011	364886	1081
1082	SF 1059	207429	Lode	MGI	09/17/2011	MGI	11/18/2011	364887	1082
1083	SF 1060	207430	Lode	MGI	09/17/2011	MGI	11/18/2011	364888	1083
1084	SF 1061	207431	Lode	MGI	09/18/2011	MGI	11/18/2011	364889	1084
1085	SF 1062	207432	Lode	MGI	09/18/2011	MGI	11/18/2011	364890	1085
1086	SF 1063	207433	Lode	MGI	09/18/2011	MGI	11/18/2011	364891	1086
1087	SF 1064	207434	Lode	MGI	09/18/2011	MGI	11/18/2011	364892	1087
1088	SF 1065	207435	Lode	MGI	09/18/2011	MGI	11/18/2011	364893	1088
1089	SF 1066	207436	Lode	MGI	09/18/2011	MGI	11/18/2011	364894	1089
1090	SF 1067	207437	Lode	MGI	09/18/2011	MGI	11/18/2011	364895	1090
1091	SF 1068	207438	Lode	MGI	09/18/2011	MGI	11/18/2011	364896	1091

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1092	SF 1069	207439	Lode	MGI	09/18/2011	MGI	11/18/2011	364897	1092
1093	SF 1070	207440	Lode	MGI	09/18/2011	MGI	11/18/2011	364898	1093
1094	SF 1071	207441	Lode	MGI	09/18/2011	MGI	11/18/2011	364899	1094
1095	SF 1072	207442	Lode	MGI	09/18/2011	MGI	11/18/2011	364900	1095
1096	SF 1073	207443	Lode	MGI	09/18/2011	MGI	11/18/2011	364901	1096
1097	SF 1074	207444	Lode	MGI	09/18/2011	MGI	11/18/2011	364902	1097
1098	SF 1075	207445	Lode	MGI	09/18/2011	MGI	11/18/2011	364903	1098
1099	SF 1076	207446	Lode	MGI	09/18/2011	MGI	11/18/2011	364904	1099
1100	SF 1077	207447	Lode	MGI	09/18/2011	MGI	11/18/2011	364905	1100
1101	SF 1078	207448	Lode	MGI	09/18/2011	MGI	11/18/2011	364906	1101
1102	SF 1079	207449	Lode	MGI	09/18/2011	MGI	11/18/2011	364907	1102
1103	SF 1080	207450	Lode	MGI	09/18/2011	MGI	11/18/2011	364908	1103
1104	SF 1081	207451	Lode	MGI	09/21/2011	MGI	11/18/2011	364909	1104
1105	SF 1082	207452	Lode	MGI	09/12/2011	MGI	11/18/2011	364910	1105
1106	SF 1083	207453	Lode	MGI	09/12/2011	MGI	11/18/2011	364911	1106
1107	SF 1084	207454	Lode	MGI	09/12/2011	MGI	11/18/2011	364912	1107
1108	SF 1085	207455	Lode	MGI	09/12/2011	MGI	11/18/2011	364913	1108
1109	SF 1086	207456	Lode	MGI	09/12/2011	MGI	11/18/2011	364914	1109
1110	SF 1087	207457	Lode	MGI	09/12/2011	MGI	11/18/2011	364915	1110
1111	SF 1088	207458	Lode	MGI	09/12/2011	MGI	11/18/2011	364916	1111
1112	SF 1089	207459	Lode	MGI	09/12/2011	MGI	11/18/2011	364917	1112
1113	SF 1090	207460	Lode	MGI	09/12/2011	MGI	11/18/2011	364918	1113
1114	SF 1091	207461	Lode	MGI	09/12/2011	MGI	11/18/2011	364919	1114
1115	SF 1092	207462	Lode	MGI	09/12/2011	MGI	11/18/2011	364920	1115
1116	SF 1093	207463	Lode	MGI	09/12/2011	MGI	11/18/2011	364921	1116
1117	SF 1094	207464	Lode	MGI	09/12/2011	MGI	11/18/2011	364922	1117
1118	SF 1095	207465	Lode	MGI	09/12/2011	MGI	11/18/2011	364923	1118
1119	SF 1096	207466	Lode	MGI	09/12/2011	MGI	11/18/2011	364924	1119

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1120	SF 1097	207467	Lode	MGI	09/12/2011	MGI	11/18/2011	364925	1120
1121	SF 1098	207468	Lode	MGI	09/12/2011	MGI	11/18/2011	364926	1121
1122	SF 1099	207469	Lode	MGI	09/13/2011	MGI	11/18/2011	364927	1122
1123	SF 1100	207470	Lode	MGI	09/17/2011	MGI	11/21/2011	364944	1123
1124	SF 1101	207471	Lode	MGI	09/13/2011	MGI	11/21/2011	364945	1124
1125	SF 1102	207472	Lode	MGI	09/13/2011	MGI	11/21/2011	364946	1125
1126	SF 1103	207473	Lode	MGI	09/13/2011	MGI	11/21/2011	364947	1126
1127	SF 1104	207474	Lode	MGI	09/13/2011	MGI	11/22/2011	364991	1127
1128	SF 1105	207475	Lode	MGI	09/13/2011	MGI	11/22/2011	364992	1128
1129	SF 1106	207476	Lode	MGI	09/13/2011	MGI	11/22/2011	364993	1129
1130	SF 1107	207477	Lode	MGI	09/13/2011	MGI	11/22/2011	364994	1130
1131	SF 1108	207478	Lode	MGI	09/13/2011	MGI	11/22/2011	364995	1131
1132	SF 1109	207479	Lode	MGI	09/13/2011	MGI	11/22/2011	364996	1132
1133	SF 1110	207480	Lode	MGI	09/13/2011	MGI	11/22/2011	364997	1133
1134	SF 1111	207481	Lode	MGI	09/13/2011	MGI	11/22/2011	364998	1134
1135	SF 1112	207482	Lode	MGI	09/13/2011	MGI	11/22/2011	364999	1135
1136	SF 1113	207483	Lode	MGI	09/13/2011	MGI	11/22/2011	365001	1136
1137	SF 1114	207484	Lode	MGI	09/13/2011	MGI	11/22/2011	365002	1137
1138	SF 1115	207485	Lode	MGI	09/13/2011	MGI	11/22/2011	365003	1138
1139	SF 1116	207486	Lode	MGI	09/13/2011	MGI	11/22/2011	365004	1139
1140	SF 1117	207487	Lode	MGI	09/13/2011	MGI	11/22/2011	365005	1140
1141	SF 1118	207488	Lode	MGI	09/13/2011	MGI	11/22/2011	365006	1141
1142	SF 1119	207489	Lode	MGI	09/12/2011	MGI	11/22/2011	365007	1142
1143	SF 1120	207490	Lode	MGI	09/12/2011	MGI	11/22/2011	365008	1143
1144	SF 1121	207491	Lode	MGI	09/12/2011	MGI	11/22/2011	365009	1144
1145	SF 1122	207492	Lode	MGI	09/12/2011	MGI	11/22/2011	365010	1145
1146	SF 1123	207493	Lode	MGI	09/12/2011	MGI	11/22/2011	365011	1146
1147	SF 1124	207494	Lode	MGI	09/12/2011	MGI	11/22/2011	365012	1147

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1148	SF 1125	207495	Lode	MGI	09/12/2011	MGI	11/22/2011	365013	1148
1149	SF 1126	207496	Lode	MGI	09/12/2011	MGI	11/22/2011	365014	1149
1150	SF 1127	207497	Lode	MGI	09/12/2011	MGI	11/22/2011	365015	1150
1151	SF 1128	207498	Lode	MGI	09/12/2011	MGI	11/22/2011	365016	1151
1152	SF 1129	207499	Lode	MGI	09/12/2011	MGI	11/22/2011	365017	1152
1153	SF 1130	207500	Lode	MGI	09/12/2011	MGI	11/22/2011	365018	1153
1154	SF 1131	207501	Lode	MGI	09/12/2011	MGI	11/22/2011	365019	1154
1155	SF 1132	207502	Lode	MGI	09/12/2011	MGI	11/22/2011	365020	1155
1156	SF 1133	207503	Lode	MGI	09/12/2011	MGI	11/22/2011	365021	1156
1157	SF 1134	207504	Lode	MGI	09/12/2011	MGI	11/22/2011	365022	1157
1158	SF 1135	207505	Lode	MGI	09/12/2011	MGI	11/22/2011	365023	1158
1159	SF 1136	207506	Lode	MGI	09/13/2011	MGI	11/22/2011	365024	1159
1160	SF 1137	207507	Lode	MGI	09/13/2011	MGI	11/22/2011	365025	1160
1161	SF 1138	207508	Lode	MGI	09/13/2011	MGI	11/22/2011	365026	1161
1162	SF 1139	207509	Lode	MGI	09/13/2011	MGI	11/22/2011	365027	1162
1163	SF 1140	207510	Lode	MGI	09/13/2011	MGI	11/22/2011	365028	1163
1164	SF 1141	207511	Lode	MGI	09/13/2011	MGI	11/22/2011	365029	1164
1165	SF 1142	207512	Lode	MGI	09/13/2011	MGI	11/22/2011	365030	1165
1166	SF 1143	207513	Lode	MGI	09/13/2011	MGI	11/22/2011	365031	1166
1167	SF 1144	207514	Lode	MGI	09/13/2011	MGI	11/22/2011	365032	1167
1168	SF 1145	207515	Lode	MGI	09/13/2011	MGI	11/22/2011	365033	1168
1169	SF 1146	207516	Lode	MGI	09/13/2011	MGI	11/22/2011	365034	1169
1170	SF 1147	207517	Lode	MGI	09/13/2011	MGI	11/22/2011	365035	1170
1171	SF 1148	207518	Lode	MGI	09/13/2011	MGI	11/22/2011	365036	1171
1172	SF 1149	207519	Lode	MGI	09/13/2011	MGI	11/22/2011	365037	1172
1173	SF 1150	207520	Lode	MGI	09/13/2011	MGI	11/22/2011	365038	1173
1174	SF 1151	207521	Lode	MGI	09/13/2011	MGI	11/22/2011	365048	1174
1175	SF 1152	207522	Lode	MGI	09/13/2011	MGI	11/22/2011	365049	1175

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1176	SF 1153	207523	Lode	MGI	09/13/2011	MGI	11/22/2011	365050	1176
1177	SF 1154	207524	Lode	MGI	09/12/2011	MGI	11/22/2011	365051	1177
1178	SF 1155	207525	Lode	MGI	09/12/2011	MGI	11/22/2011	365052	1178
1179	SF 1156	207526	Lode	MGI	09/12/2011	MGI	11/22/2011	365053	1179
1180	SF 1157	207527	Lode	MGI	09/12/2011	MGI	11/22/2011	365054	1180
1181	SF 1158	207528	Lode	MGI	09/12/2011	MGI	11/22/2011	365055	1181
1182	SF 1159	207529	Lode	MGI	09/12/2011	MGI	11/22/2011	365056	1182
1183	SF 1160	207530	Lode	MGI	09/12/2011	MGI	11/22/2011	365057	1183
1184	SF 1161	207531	Lode	MGI	09/12/2011	MGI	11/22/2011	365058	1184
1185	SF 1162	207532	Lode	MGI	09/12/2011	MGI	11/22/2011	365059	1185
1186	SF 1163	207533	Lode	MGI	09/12/2011	MGI	11/22/2011	365060	1186
1187	SF 1164	207534	Lode	MGI	09/12/2011	MGI	11/22/2011	365061	1187
1188	SF 1165	207535	Lode	MGI	09/12/2011	MGI	11/22/2011	365062	1188
1189	SF 1166	207536	Lode	MGI	09/12/2011	MGI	11/22/2011	365063	1189
1190	SF 1167	207537	Lode	MGI	09/12/2011	MGI	11/22/2011	365064	1190
1191	SF 1168	207538	Lode	MGI	09/12/2011	MGI	11/22/2011	365065	1191
1192	SF 1169	207539	Lode	MGI	09/12/2011	MGI	11/22/2011	365066	1192
1193	SF 1170	207540	Lode	MGI	09/12/2011	MGI	11/22/2011	365067	1193
1194	SF 1171	207541	Lode	MGI	09/13/2011	MGI	11/22/2011	365068	1194
1195	SF 1172	207542	Lode	MGI	09/13/2011	MGI	11/22/2011	365069	1195
1196	SF 1173	207543	Lode	MGI	09/13/2011	MGI	11/22/2011	365070	1196
1197	SF 1174	207544	Lode	MGI	09/13/2011	MGI	11/22/2011	365071	1197
1198	SF 1175	207545	Lode	MGI	09/13/2011	MGI	11/22/2011	365072	1198
1199	SF 1176	207546	Lode	MGI	09/13/2011	MGI	11/22/2011	365073	1199
1200	SF 1177	207547	Lode	MGI	09/13/2011	MGI	11/22/2011	365074	1200
1201	SF 1178	207548	Lode	MGI	09/13/2011	MGI	11/22/2011	365075	1201
1202	SF 1179	207549	Lode	MGI	09/13/2011	MGI	11/22/2011	365076	1202
1203	SF 1180	207550	Lode	MGI	09/13/2011	MGI	11/22/2011	365077	1203

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1204	SF 1181	207551	Lode	MGI	09/18/2011	MGI	11/22/2011	365078	1204
1205	SF 1182	207552	Lode	MGI	09/18/2011	MGI	11/22/2011	365079	1205
1206	SF 1183	207553	Lode	MGI	09/18/2011	MGI	11/22/2011	365080	1206
1207	SF 1184	207554	Lode	MGI	09/18/2011	MGI	11/22/2011	365081	1207
1208	SF 1185	207555	Lode	MGI	09/18/2011	MGI	11/22/2011	365082	1208
1209	SF 1186	207556	Lode	MGI	09/18/2011	MGI	11/22/2011	365083	1209
1210	SF 1187	207557	Lode	MGI	09/18/2011	MGI	11/22/2011	365084	1210
1211	SF 1188	207558	Lode	MGI	09/18/2011	MGI	11/22/2011	365085	1211
1212	SF 1189	207559	Lode	MGI	09/18/2011	MGI	11/22/2011	365086	1212
1213	SF 1190	207560	Lode	MGI	09/18/2011	MGI	11/22/2011	365087	1213
1214	SF 1191	207561	Lode	MGI	09/18/2011	MGI	11/22/2011	365088	1214
1215	SF 1192	207562	Lode	MGI	09/18/2011	MGI	11/22/2011	365089	1215
1216	SF 1193	207563	Lode	MGI	09/18/2011	MGI	11/22/2011	365090	1216
1217	SF 1194	207564	Lode	MGI	09/18/2011	MGI	11/22/2011	365091	1217
1218	SF 1195	207565	Lode	MGI	09/18/2011	MGI	11/22/2011	365092	1218
1219	SF 1196	207566	Lode	MGI	09/18/2011	MGI	11/22/2011	365093	1219
1220	SF 1197	207567	Lode	MGI	09/18/2011	MGI	11/22/2011	365094	1220
1221	SF 1198	207568	Lode	MGI	09/18/2011	MGI	11/22/2011	365095	1221
1222	SF 1199	207569	Lode	MGI	09/18/2011	MGI	11/22/2011	365096	1222
1223	SF 1200	207570	Lode	MGI	09/21/2011	MGI	11/22/2011	365098	1223
1224	SF 1201	207571	Lode	MGI	09/21/2011	MGI	11/22/2011	365099	1224
1225	SF 1202	207572	Lode	MGI	09/21/2011	MGI	11/22/2011	365100	1225
1226	SF 1203	207573	Lode	MGI	09/21/2011	MGI	11/22/2011	365101	1226
1227	SF 1204	207574	Lode	MGI	09/18/2011	MGI	11/22/2011	365102	1227
1228	SF 1205	207575	Lode	MGI	09/18/2011	MGI	11/22/2011	365103	1228
1229	SF 1206	207576	Lode	MGI	09/18/2011	MGI	11/22/2011	365104	1229
1230	SF 1207	207577	Lode	MGI	09/18/2011	MGI	11/22/2011	365105	1230
1231	SF 1208	207578	Lode	MGI	09/18/2011	MGI	11/22/2011	365106	1231

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1232	SF 1209	207579	Lode	MGI	09/18/2011	MGI	11/22/2011	365107	1232
1233	SF 1210	207580	Lode	MGI	09/18/2011	MGI	11/22/2011	365108	1233
1234	SF 1211	207581	Lode	MGI	09/17/2011	MGI	11/22/2011	365109	1234
1235	SF 1212	207582	Lode	MGI	09/18/2011	MGI	11/22/2011	365110	1235
1236	SF 1213	207583	Lode	MGI	09/18/2011	MGI	11/22/2011	365111	1236
1237	SF 1214	207584	Lode	MGI	09/18/2011	MGI	11/22/2011	365112	1237
1238	SF 1215	207585	Lode	MGI	09/18/2011	MGI	11/22/2011	365113	1238
1239	SF 1216	207586	Lode	MGI	09/18/2011	MGI	11/22/2011	365114	1239
1240	SF 1217	207587	Lode	MGI	09/18/2011	MGI	11/22/2011	365115	1240
1241	SF 1218	207588	Lode	MGI	09/18/2011	MGI	11/22/2011	365116	1241
1242	SF 1219	207589	Lode	MGI	09/18/2011	MGI	11/22/2011	365117	1242
1243	SF 1220	207590	Lode	MGI	09/21/2011	MGI	11/22/2011	365118	1243
1244	SF 1221	207591	Lode	MGI	09/21/2011	MGI	11/22/2011	365119	1244
1245	SF 1222	207592	Lode	MGI	09/21/2011	MGI	11/22/2011	365120	1245
1246	SF 1223	207593	Lode	MGI	09/21/2011	MGI	11/22/2011	365121	1246
1247	SF 1224	207594	Lode	MGI	09/21/2011	MGI	11/22/2011	365122	1247
1248	SF 1225	207595	Lode	MGI	09/21/2011	MGI	11/22/2011	365123	1248
1249	SF 1226	207596	Lode	MGI	09/21/2011	MGI	11/22/2011	365124	1249
1250	SF 1227	207597	Lode	MGI	09/21/2011	MGI	11/22/2011	365125	1250
1251	SF 1228	207598	Lode	MGI	09/21/2011	MGI	11/22/2011	365126	1251
1252	SF 1229	207599	Lode	MGI	09/21/2011	MGI	11/22/2011	365127	1252
1253	SF 1230	207600	Lode	MGI	09/21/2011	MGI	11/22/2011	365128	1253
1254	SF 1231	207601	Lode	MGI	09/21/2011	MGI	11/22/2011	365129	1254
1255	SF 1232	207602	Lode	MGI	09/21/2011	MGI	11/22/2011	365130	1255
1256	SF 1233	207603	Lode	MGI	09/21/2011	MGI	11/22/2011	365131	1256
1257	SF 1234	207604	Lode	MGI	09/21/2011	MGI	11/22/2011	365132	1257
1258	SF 1235	207605	Lode	MGI	09/21/2011	MGI	11/22/2011	365133	1258
1259	SF 1236	207606	Lode	MGI	09/21/2011	MGI	11/22/2011	365134	1259

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1260	SF 1237	207607	Lode	MGI	09/21/2011	MGI	11/22/2011	365135	1260
1261	SF 1238	207608	Lode	MGI	09/21/2011	MGI	11/22/2011	365136	1261
1262	SF 1239	207609	Lode	MGI	09/21/2011	MGI	11/22/2011	365137	1262
1263	SF 1240	207610	Lode	MGI	09/21/2011	MGI	11/22/2011	365138	1263
1264	SF 1241	207611	Lode	MGI	09/21/2011	MGI	11/22/2011	365139	1264
1265	SF 1242	207612	Lode	MGI	09/21/2011	MGI	11/22/2011	365140	1265
1266	SF 1243	207613	Lode	MGI	09/21/2011	MGI	11/22/2011	365141	1266
1267	SF 1244	207614	Lode	MGI	09/18/2011	MGI	11/22/2011	365142	1267
1268	SF 1245	207615	Lode	MGI	09/21/2011	MGI	11/22/2011	365143	1268
1269	SF 1246	207616	Lode	MGI	09/21/2011	MGI	11/22/2011	365144	1269
1270	SF 1247	207617	Lode	MGI	09/21/2011	MGI	11/22/2011	365145	1270
1271	SF 1248	207618	Lode	MGI	09/21/2011	MGI	11/22/2011	365146	1271
1272	SF 1249	207619	Lode	MGI	09/21/2011	MGI	11/22/2011	365147	1272
1273	SF 1250	207620	Lode	MGI	09/21/2011	MGI	11/22/2011	365148	1273
1274	SF 1251	207621	Lode	MGI	09/21/2011	MGI	11/22/2011	365149	1274
1275	SF 1252	207622	Lode	MGI	09/21/2011	MGI	11/22/2011	365150	1275
1276	SF 1253	207623	Lode	MGI	09/21/2011	MGI	11/22/2011	365151	1276
1277	SF 1254	207624	Lode	MGI	09/21/2011	MGI	11/22/2011	365152	1277
1278	SF 1255	207625	Lode	MGI	09/21/2011	MGI	11/22/2011	365153	1278
1279	SF 1256	207626	Lode	MGI	09/21/2011	MGI	11/22/2011	365154	1279
1280	SF 1257	207627	Lode	MGI	09/21/2011	MGI	11/22/2011	365155	1280
1281	SF 1258	207628	Lode	MGI	09/21/2011	MGI	11/22/2011	365156	1281
1282	SF 1259	207629	Lode	MGI	09/21/2011	MGI	11/22/2011	365157	1282
1283	SF 1260	207630	Lode	MGI	09/21/2011	MGI	11/22/2011	365158	1283
1284	SF 1261	207631	Lode	MGI	09/21/2011	MGI	11/22/2011	365159	1284
1285	SF 1262	207632	Lode	MGI	09/20/2011	MGI	11/22/2011	365160	1285
1286	SF 1263	207633	Lode	MGI	09/20/2011	MGI	11/22/2011	365161	1286
1287	SF 1264	207634	Lode	MGI	09/20/2011	MGI	11/22/2011	365168	1287

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1288	SF 1265	207635	Lode	MGI	09/20/2011	MGI	11/22/2011	365169	1288
1289	SF 1266	207636	Lode	MGI	09/20/2011	MGI	11/22/2011	365170	1289
1290	SF 1267	207637	Lode	MGI	09/20/2011	MGI	11/22/2011	365171	1290
1291	SF 1268	207638	Lode	MGI	09/20/2011	MGI	11/22/2011	365172	1291
1292	SF 1269	207639	Lode	MGI	09/20/2011	MGI	11/22/2011	365173	1292
1293	SF 1270	207640	Lode	MGI	09/20/2011	MGI	11/22/2011	365174	1293
1294	SF 1271	207641	Lode	MGI	09/20/2011	MGI	11/22/2011	365175	1294
1295	SF 1272	207642	Lode	MGI	09/20/2011	MGI	11/22/2011	365176	1295
1296	SF 1273	207643	Lode	MGI	09/20/2011	MGI	11/22/2011	365177	1296
1297	SF 1274	207644	Lode	MGI	09/20/2011	MGI	11/22/2011	365178	1297
1298	SF 1275	207645	Lode	MGI	09/20/2011	MGI	11/22/2011	365179	1298
1299	SF 1276	207646	Lode	MGI	09/20/2011	MGI	11/22/2011	365180	1299
1300	SF 1277	207647	Lode	MGI	09/20/2011	MGI	11/22/2011	365181	1300
1301	SF 1278	207648	Lode	MGI	09/20/2011	MGI	11/22/2011	365182	1301
1302	SF 1279	207649	Lode	MGI	09/20/2011	MGI	11/22/2011	365183	1302
1303	SF 1280	207650	Lode	MGI	09/20/2011	MGI	11/22/2011	365184	1303
1304	SF 1281	207651	Lode	MGI	09/20/2011	MGI	11/22/2011	365185	1304
1305	SF 1282	207652	Lode	MGI	09/20/2011	MGI	11/22/2011	365186	1305
1306	SF 1283	207653	Lode	MGI	09/20/2011	MGI	11/22/2011	365187	1306
1307	SF 1284	207654	Lode	MGI	09/20/2011	MGI	11/22/2011	365188	1307
1308	SF 1285	207655	Lode	MGI	09/20/2011	MGI	11/22/2011	365189	1308
1309	SF 1286	207656	Lode	MGI	09/20/2011	MGI	11/22/2011	365190	1309
1310	SF 1287	207657	Lode	MGI	09/20/2011	MGI	11/22/2011	365191	1310
1311	SF 1288	207658	Lode	MGI	09/20/2011	MGI	11/22/2011	365192	1311
1312	SF 1289	207659	Lode	MGI	09/20/2011	MGI	11/22/2011	365193	1312
1313	SF 1290	207660	Lode	MGI	09/20/2011	MGI	11/22/2011	365194	1313
1314	SF 1291	207661	Lode	MGI	09/20/2011	MGI	11/22/2011	365195	1314
1315	SF 1292	207662	Lode	MGI	09/20/2011	MGI	11/22/2011	365196	1315

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1316	SF 1293	207663	Lode	MGI	09/20/2011	MGI	11/22/2011	365197	1316
1317	SF 1294	207664	Lode	MGI	09/20/2011	MGI	11/22/2011	365198	1317
1318	SF 1295	207665	Lode	MGI	09/20/2011	MGI	11/22/2011	365199	1318
1319	SF 1296	207666	Lode	MGI	09/20/2011	MGI	11/22/2011	365200	1319
1320	SF 1297	207667	Lode	MGI	09/20/2011	MGI	11/22/2011	365201	1320
1321	SF 1298	207668	Lode	MGI	09/20/2011	MGI	11/22/2011	365202	1321
1322	SF 1299	207669	Lode	MGI	09/20/2011	MGI	11/22/2011	365203	1322
1323	SF 1300	207670	Lode	MGI	09/20/2011	MGI	11/23/2011	365209	1323
1324	SF 1301	207671	Lode	MGI	09/20/2011	MGI	11/23/2011	365210	1324
1325	SF 1302	207672	Lode	MGI	09/20/2011	MGI	11/23/2011	365211	1325
1326	SF 1303	207673	Lode	MGI	09/19/2011	MGI	11/23/2011	365212	1326
1327	SF 1304	207674	Lode	MGI	09/19/2011	MGI	11/23/2011	365213	1327
1328	SF 1305	207675	Lode	MGI	09/19/2011	MGI	11/23/2011	365214	1328
1329	SF 1306	207676	Lode	MGI	09/19/2011	MGI	11/23/2011	365215	1329
1330	SF 1307	207677	Lode	MGI	09/19/2011	MGI	11/23/2011	365216	1330
1331	SF 1308	207678	Lode	MGI	09/19/2011	MGI	11/23/2011	365217	1331
1332	SF 1309	207679	Lode	MGI	09/19/2011	MGI	11/23/2011	365218	1332
1333	SF 1310	207680	Lode	MGI	09/19/2011	MGI	11/23/2011	365219	1333
1334	SF 1311	207681	Lode	MGI	09/19/2011	MGI	11/23/2011	365220	1334
1335	SF 1312	207682	Lode	MGI	09/19/2011	MGI	11/23/2011	365221	1335
1336	SF 1313	207683	Lode	MGI	09/19/2011	MGI	11/23/2011	365222	1336
1337	SF 1314	207684	Lode	MGI	09/19/2011	MGI	11/23/2011	365223	1337
1338	SF 1315	207685	Lode	MGI	09/19/2011	MGI	11/23/2011	365227	1338
1339	SF 1316	207686	Lode	MGI	09/19/2011	MGI	11/23/2011	365228	1339
1340	SF 1317	207687	Lode	MGI	09/19/2011	MGI	11/23/2011	365229	1340
1341	SF 1318	207688	Lode	MGI	09/19/2011	MGI	11/23/2011	365230	1341
1342	SF 1319	207689	Lode	MGI	09/19/2011	MGI	11/23/2011	365231	1342
1343	SF 1320	207690	Lode	MGI	09/19/2011	MGI	11/23/2011	365232	1343

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1344	SF 1321	207691	Lode	MGI	09/19/2011	MGI	11/23/2011	365233	1344
1345	SF 1322	207692	Lode	MGI	09/19/2011	MGI	11/23/2011	365234	1345
1346	SF 1323	207693	Lode	MGI	09/19/2011	MGI	11/23/2011	365238	1346
1347	SF 1324	207694	Lode	MGI	09/19/2011	MGI	11/23/2011	365239	1347
1348	SF 1325	207695	Lode	MGI	09/19/2011	MGI	11/23/2011	365240	1348
1349	SF 1326	207696	Lode	MGI	09/19/2011	MGI	11/23/2011	365241	1349
1350	SF 1327	207697	Lode	MGI	09/19/2011	MGI	11/23/2011	365242	1350
1351	SF 1328	207698	Lode	MGI	09/19/2011	MGI	11/23/2011	365243	1351
1352	SF 1329	207699	Lode	MGI	09/19/2011	MGI	11/23/2011	365244	1352
1353	SF 1330	207700	Lode	MGI	09/19/2011	MGI	11/23/2011	365245	1353
1354	SF 1331	207701	Lode	MGI	09/19/2011	MGI	11/23/2011	365246	1354
1355	SF 1332	207702	Lode	MGI	09/19/2011	MGI	11/23/2011	365248	1355
1356	SF 1333	207703	Lode	MGI	09/19/2011	MGI	11/23/2011	365249	1356
1357	SF 1334	207704	Lode	MGI	09/19/2011	MGI	11/23/2011	365250	1357
1358	SF 1335	207705	Lode	MGI	09/19/2011	MGI	11/23/2011	365251	1358
1359	SF 1336	207706	Lode	MGI	09/19/2011	MGI	11/23/2011	365252	1359
1360	SF 1337	207707	Lode	MGI	09/19/2011	MGI	11/23/2011	365253	1360
1361	SF 1338	207708	Lode	MGI	09/19/2011	MGI	11/23/2011	365254	1361
1362	SF 1339	207709	Lode	MGI	09/19/2011	MGI	11/23/2011	365255	1362
1363	SF 1340	207710	Lode	MGI	09/19/2011	MGI	11/23/2011	365256	1363
1364	SF 1341	207711	Lode	MGI	09/19/2011	MGI	11/23/2011	365257	1364
1365	SF 1342	207712	Lode	MGI	09/19/2011	MGI	11/23/2011	365258	1365
1366	SF 1343	207713	Lode	MGI	09/19/2011	MGI	11/23/2011	365259	1366
1367	SF 1344	207714	Lode	MGI	09/19/2011	MGI	11/23/2011	365260	1367
1368	SF 1345	207715	Lode	MGI	09/19/2011	MGI	11/23/2011	365261	1368
1369	SF 1346	207716	Lode	MGI	09/19/2011	MGI	11/23/2011	365262	1369
1370	SF 1347	207717	Lode	MGI	09/19/2011	MGI	11/23/2011	365269	1370
1371	SF 1348	207718	Lode	MGI	09/19/2011	MGI	11/23/2011	365270	1371

Count	Claim Name	IMC No.[1]	Claim Type	Owner[2]	Location Date	Locator[3]	Recorded Date	Instrument No.[4]	Count
1372	SF 1349	207719	Lode	MGI	09/19/2011	MGI	11/23/2011	365271	1372
1373	SF 1350	207720	Lode	MGI	09/19/2011	MGI	11/23/2011	365272	1373
1374	SF 1351	207721	Lode	MGI	09/19/2011	MGI	11/23/2011	365273	1374
1375	SF 1352	207722	Lode	MGI	09/19/2011	MGI	11/23/2011	365274	1375
1376	SF 1353	207723	Lode	MGI	09/19/2011	MGI	11/23/2011	365275	1376
1377	SF 1354	207724	Lode	MGI	09/19/2011	MGI	11/23/2011	365276	1377
1378	SF 1355	207725	Lode	MGI	09/20/2011	MGI	11/23/2011	365277	1378
1379	SF 1356	207145	Lode	MGI	10/05/2011	MGI	10/31/2011	363890	1379
1380	SF 1357	207146	Lode	MGI	10/05/2011	MGI	10/31/2011	363891	1380
1381	SF 1358	207147	Lode	MGI	10/05/2011	MGI	10/31/2011	363892	1381
1382	YP1	186740	Lode	IGR	10/15/2003	Vista	12/02/2003	278442	1382
1383	YP2	186741	Lode	IGR	10/15/2003	Vista	12/02/2003	278443	1383
1384	YP3	186742	Lode	IGR	10/15/2003	Vista	12/02/2003	278444	1384
1385	YP4	186743	Lode	IGR	10/15/2003	Vista	12/02/2003	278445	1385
1386	YP5	186744	Lode	IGR	10/15/2003	Vista	12/02/2003	278446	1386
1387	YP6	186745	Lode	IGR	10/15/2003	Vista	12/02/2003	278447	1387
1388	YP7	186746	Lode	IGR	10/19/2003	Vista	12/02/2003	278448	1388
1389	YP8	186747	Lode	IGR	10/19/2003	Vista	12/02/2003	278449	1389

Notes:
1. *IMC = Bureau of Land Management (BLM) Recordation Serial Number.*
2. *MGI = Midas Gold, Inc., IGR = Idaho Gold Resources LLC.*
3. *MGI = Midas Gold, Inc., MGIAC = MGI Acquisition Corp., Niagara = Niagara Mining and Development Co., Inc., Vista = Vista Gold Corp.*
4. *Certificate of Location recorded in Valley County, Idaho with listed Instrument Number.*

APPENDIX III: FINANCIAL MODEL

Mining and Processing
All Cases

Stibnite Gold Project
Midas Gold Corporation

		Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Mine															
Yellow Pine															
High Antimony	kst	6,750	225	1,358	1,310	666	1,688	748	519	236					
Gold grade	oz/st	0.065	0.061	0.072	0.073	0.062	0.055	0.058	0.066	0.085					
Silver grade	oz/st	0.21	0.25	0.21	0.22	0.17	0.26	0.15	0.17	0.12					
Antimony grade	%	0.59%	0.66%	0.52%	0.60%	0.58%	0.71%	0.52%	0.68%	0.22%					
Contained Gold	kozs	438	14	97	95	41	94	43	34	20					
Contained Silver	kozs	1,420	56	292	291	112	442	112	88	29					
Contained Antimony	klbs	80,011	2,951	14,098	15,591	7,699	23,902	7,734	7,017	1,020					
Low Antimony	kst	37,235	213	3,133	5,963	6,346	5,542	6,308	5,749	3,981					
Gold grade	oz/t	0.056	0.046	0.066	0.062	0.052	0.053	0.052	0.053	0.061					
Silver grade	oz/t	0.07	0.15	0.13	0.08	0.06	0.07	0.06	0.05	0.05					
Antimony grade	%	0.01%	0.01%	0.02%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%					
Contained Gold	kozs	2,085	10	206	371	329	292	327	306	244					
Contained Silver	kozs	2,553	31	407	503	349	405	366	305	187					
Contained Antimony	klbs	6,365	55	1,035	1,196	888	887	1,135	690	478					
Total Yellow Pine	kst	43,985	438	4,491	7,273	7,012	7,230	7,056	6,268	4,217					
Gold grade	oz/st	0.057	0.054	0.067	0.064	0.053	0.053	0.053	0.054	0.063					
Silver grade	oz/st	0.09	0.20	0.16	0.11	0.07	0.12	0.07	0.06	0.05					
Antimony grade	%	0.10%	0.34%	0.17%	0.12%	0.06%	0.17%	0.06%	0.06%	0.02%					
Contained Gold	kozs	2,523	23	303	466	371	386	371	340	264					
Contained Silver	kozs	3,973	87	699	794	461	847	478	392	216					
Contained Antimony	klbs	86,376	3,007	15,132	16,787	8,587	24,789	8,870	7,707	1,497					
Waste Rock	kst	124,304	5,544	10,343	19,712	27,751	31,457	18,945	8,249	2,303					
Hangar Flats															
High Antimony	kst	4,284			3	27	10		137	736	2,755	500	116		
Gold grade	oz/st	0.056			0.044	0.033	0.029		0.061	0.051	0.059	0.045	0.046		
Silver grade	oz/st	0.17			0.06	0.06	0.06		0.16	0.13	0.20	0.09	0.12		
Antimony grade	%	0.43%			0.11%	0.13%	0.11%		0.46%	0.45%	0.47%	0.19%	0.26%		
Contained Gold	kozs	238			0.1	0.9	0.3		8.3	37.8	162.5	22.7	5.3		
Contained Silver	kozs	712			0.2	1.7	0.6		21.2	92.7	537.2	44.5	13.8		
Contained Antimony	klbs	36,438			6	71	22		1,255	6,653	25,897	1,940	594		
Low Antimony	kst	11,146			113	351	150	336	722	1,963	3,626	2,625	1,260		
Gold grade	oz/st	0.040			0.032	0.033	0.027	0.027	0.053	0.039	0.043	0.038	0.043		
Silver grade	oz/st	0.06			0.05	0.04	0.03	0.03	0.06	0.06	0.07	0.04	0.06		
Antimony grade	%	0.02%			0.02%	0.01%	0.02%	0.01%	0.02%	0.02%	0.02%	0.02%	0.02%		
Contained Gold	kozs	449			3.6	11.6	4.0	9.1	38.4	75.8	154.1	99.0	53.7		
Contained Silver	kozs	615			5.3	14.0	5.1	9.7	46.2	117.8	235.7	110.3	70.6		
Contained Antimony	klbs	4,319			48	84	72	40	332	746	1,595	998	403		
Total Hangar Flats	kst	15,430			116	378	160	336	859	2,699	6,381	3,125	1,376		
Gold grade	oz/st	0.045			0.032	0.033	0.027	0.027	0.054	0.042	0.050	0.039	0.043		
Silver grade	oz/st	0.09			0.05	0.04	0.04	0.03	0.08	0.08	0.12	0.05	0.06		
Antimony grade	%	0.13%			0.02%	0.02%	0.03%	0.01%	0.09%	0.14%	0.22%	0.05%	0.04%		
Contained Gold	kozs	687			3.7	12.5	4.3	9.1	46.8	113.6	316.7	121.7	59.0		
Contained Silver	kozs	1,327			5.5	15.7	5.7	9.7	67.4	210.5	772.9	154.8	84.4		
Contained Antimony	klbs	40,757			55	155	94	40	1,587	7,399	27,492	2,938	997		
Waste Rock		86,696	2,246	-	2,190	3,679	1,547	9,900	10,659	21,005	26,247	7,479	1,744		

Mining and Processing
All Cases

Stibnite Gold Project
Midas Gold Corporation

		Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
West End															
Low Antimony	kst	24,914							249	453	904	3,652	5,094	7,245	7,317
Gold grade	oz/st	0.041							0.038	0.057	0.048	0.041	0.037	0.041	0.043
Silver grade	oz/st	0.04							0.02	0.02	0.02	0.03	0.05	0.06	0.05
Contained Gold	kozs	1,024							9.6	26.0	42.9	148.6	185.9	299.2	311.7
Contained Silver	kozs	1,090							5.5	7.7	18.1	91.3	239.4	398.5	329.3
Oxide Feed	kt	10,736	129	1,782	660	660	660	658	674	681	765	1,273	1,580	805	409
Gold grade	oz/st	0.022	0.033	0.022	0.024	0.022	0.021	0.021	0.016	0.019	0.018	0.023	0.023	0.025	0.023
Silver grade	oz/st	0.03	0.05	0.02	0.02	0.03	0.03	0.02	0.01	0.02	0.02	0.03	0.04	0.05	0.05
Contained Gold	kozs	233	4.3	39.5	15.5	14.5	13.9	13.8	11.1	12.7	13.8	28.6	36.5	19.8	9.3
Contained Silver	kozs	314	6.6	41.2	16.2	17.2	17.2	13.2	9.4	15.7	15.3	39.5	63.2	41.1	18.4
Total West End	kst	35,650	129	1,782	660	660	660	658	923	1,134	1,669	4,925	6,674	8,050	7,726
Gold grade	oz/st	0.035	0.033	0.022	0.024	0.022	0.021	0.021	0.022	0.034	0.034	0.036	0.033	0.040	0.042
Silver grade	oz/st	0.04	0.05	0.02	0.02	0.03	0.03	0.02	0.02	0.02	0.02	0.03	0.05	0.05	0.05
Contained Gold	kozs	1,257	4.3	39.5	15.5	14.5	13.9	13.8	20.6	38.7	56.8	177.3	222.4	319.0	321.0
Contained Silver	kozs	1,404	6.6	41.2	16.2	17.2	17.2	13.2	14.9	23.4	33.4	130.8	302.6	439.5	347.7
Waste Rock		125,217	-	830	5,606	1,508	1,374	946	5,105	15,042	10,642	7,703	26,470	31,277	18,714
Historic Tailings	kst	3,001		477	916	916	692								
Gold grade	oz/st	0.034		0.034	0.031	0.037	0.032								
Silver grade	oz/st	0.08		0.08	0.07	0.08	0.11								
Contained Gold	kozs	101		16.4	28.9	33.9	22.1								
Contained Silver	kozs	252		37.5	60.9	76.0	77.5								
Spent Ore and Inferred Historic Tailings		5,752	5,752												
Process Plant															
Yellow Pine															
High Antimony	kst	6,750		1,583	1,310	666	1,688	748	519	236					
Gold grade	oz/st	0.065		0.070	0.073	0.062	0.055	0.058	0.066	0.085					
Silver grade	oz/st	0.21		0.22	0.22	0.17	0.26	0.15	0.17	0.12					
Antimony grade	%	0.59%		0.54%	0.60%	0.58%	0.71%	0.52%	0.68%	0.22%					
Contained Gold	kozs	438		111	95	41	94	43	34	20					
Contained Silver	kozs	1,420		347	291	112	442	112	88	29					
Contained Antimony	klbs	80,011		17,049	15,591	7,699	23,902	7,734	7,017	1,020					
Gold Bullion Recovery	%	87.2%		87.8%	87.9%	86.8%	86.0%	86.3%	87.3%	89.5%					
Silver Bullion Recovery	%	8.5%		8.5%	8.5%	8.5%	8.5%	8.5%	8.5%	8.5%					
Recovered Gold	kozs	382		97.4	83.6	35.7	80.4	37.2	29.7	17.9					
Recovered Silver	kozs	121		29.5	24.7	9.5	37.6	9.5	7.5	2.4					
Antimony - Gold Recovery	%	2.7%		2.60%	2.66%	2.69%	2.89%	2.72%	2.69%	2.04%					
Antimony - Silver Recovery	%	43.0%		43.0%	43.0%	42.9%	43.1%	42.9%	43.1%	43.1%					
Antimony Recovery	%	87.3%		86.7%	87.1%	87.1%	88.4%	86.0%	88.1%	77.6%					
Antimony Concentrate	kst	59.2		12.5	11.5	5.7	17.9	5.6	5.2	0.7					
Antimony Concentrate Grade	%	59%		59%	59%	59%	59%	59%	59%	59%					
Recovered Gold	kozs	12		2.9	2.5	1.1	2.7	1.2	0.9	0.4					
Recovered Silver	kozs	611		149	125	48	191	48	38	12					
Recovered Antimony	klbs	69,822		14,776	13,584	6,705	21,137	6,648	6,180	792					
Low Antimony	kst	37,235		3,346	5,963	6,346	5,542	6,308	5,749	3,981					
Gold grade	oz/st	0.056		0.064	0.062	0.052	0.053	0.052	0.053	0.061					
Silver grade	oz/st	0.07		0.13	0.08	0.06	0.07	0.06	0.05	0.05					
Antimony grade	%	0.01%		0.02%	0.01%	0.01%	0.01%	0.01%	0.01%	0.01%					
Contained Gold	kozs	2,085		216	371	329	292	327	306	244					
Contained Silver	kozs	2,553		438	503	349	405	366	305	187					
Contained Antimony	klbs	6,365		1,090	1,196	888	887	1,135	690	478					
Gold Bullion Recovery	%	90.2%		90.6%	90.6%	89.7%	90.2%	90.1%	89.9%	90.6%					
Silver Bullion Recovery	%	8.5%		8.5%	8.5%	8.5%	8.5%	8.5%	8.5%	8.5%					
Recovered Gold	kozs	1,881		195	336	296	264	295	275	221					
Recovered Silver	kozs	217		37	43	30	34	31	26	16					

Stibnite Gold Project
Midas Gold Corporation

		Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Total Yellow Pine	kst	43,985		4,929	7,273	7,012	7,230	7,056	6,268	4,217					
Gold grade	oz/st	0.057		0.066	0.064	0.053	0.053	0.053	0.054	0.063					
Silver grade	oz/st	0.09		0.16	0.11	0.07	0.12	0.07	0.06	0.05					
Antimony grade	%	0.10%		0.18%	0.12%	0.06%	0.17%	0.06%	0.06%	0.02%					
Contained Gold	kozs	2,523		327	466	371	386	371	340	264					
Contained Silver	kozs	3,973		785	794	461	847	478	392	216					
Contained Antimony	klbs	86,376		18,139	16,787	8,587	24,789	8,870	7,707	1,497					
Gold Bullion Recovery	%	89.7%		89.6%	90.0%	89.4%	89.2%	89.7%	89.7%	90.5%					
Silver Bullion Recovery	%	8.5%		8.5%	8.5%	8.5%	8.5%	8.5%	8.5%	8.5%					
Recovered Gold	kozs	2,263		293	419	331	344	332	305	239					
Recovered Silver	kozs	338		67	67	39	72	41	33	18					
Antimony - Gold Recovery	%	0.5%		0.89%	0.54%	0.30%	0.70%	0.32%	0.27%	0.15%					
Antimony - Silver Recovery	%	15.4%		19.0%	15.8%	10.4%	22.5%	10.1%	9.6%	5.7%					
Antimony Recovery	%	80.8%		81.5%	80.9%	78.1%	85.3%	75.0%	80.2%	52.9%					
Antimony Concentrate	kst	59		12.5	11.5	5.7	17.9	5.6	5.2	0.7					
Antimony Concentrate Grade	%	59%		59%	59%	59%	59%	59%	59%	59%					
Recovered Gold	kozs	12		2.89	2.53	1.11	2.70	1.17	0.91	0.41					
Recovered Silver	kozs	611		149	125	48	191	48	38	12					
Recovered Antimony	klbs	69,822		14,776	13,584	6,705	21,137	6,648	6,180	792					
Hangar Flats															
High Antimony	kst	4,284			3	27	10		137	736	2,755	500	116		
Gold grade	oz/st	0.056			0.044	0.033	0.029		0.061	0.051	0.059	0.045	0.046		
Silver grade	oz/st	0.17			0.06	0.06	0.06		0.16	0.13	0.20	0.09	0.12		
Antimony grade	%	0.43%			0.11%	0.13%	0.11%		0.46%	0.45%	0.47%	0.19%	0.26%		
Contained Gold	kozs	238			0.13	0.89	0.29		8.34	37.83	162.55	22.70	5.34		
Contained Silver	kozs	712			0.17	1.67	0.59		21.24	92.74	537.23	44.50	13.80		
Contained Antimony	klbs	36,438			6	71	22		1,255	6,653	25,897	1,940	594		
Gold Bullion Recovery	%	84.0%			89.5%	88.7%	86.1%		83.5%	83.9%	83.6%	86.5%	86.9%		
Silver Bullion Recovery	%	5.1%			5.1%	5.1%	5.0%		5.1%	5.1%	5.1%	5.1%	5.1%		
Recovered Gold	kozs	200			0.12	0.79	0.25		6.97	31.75	135.83	19.64	4.64		
Recovered Silver	kozs	36			0.01	0.08	0.03		1.08	4.75	27.40	2.25	0.70		
Antimony - Gold Recovery	%	2.2%			1.30%	1.03%	0.85%		2.17%	1.93%	2.41%	1.78%	1.80%		
Antimony - Silver Recovery	%	49.1%			48.7%	49.4%	49.0%		49.1%	49.1%	49.1%	48.8%	49.0%		
Antimony Recovery	%	82.4%			71.7%	73.5%	72.2%		83.0%	82.6%	82.9%	76.3%	78.5%		
Antimony Concentrate	kt	25			0.00	0.04	0.01		0.88	4.66	18.20	1.26	0.40		
Antimony Concentrate Grade	%	59%			59%	59%	59%		59%	59%	59%	59%	59%		
Recovered Gold	kozs	5			0.002	0.009	0.003		0.181	0.729	3.912	0.405	0.096		
Recovered Silver	kozs	349			0.08	0.83	0.29		10.43	45.56	263.65	21.70	6.76		
Recovered Antimony	klbs	30,030			5	52	16		1,041	5,496	21,472	1,481	466		
Low Antimony	kst	11,146			113	351	150	336	722	1,963	3,626	2,625	1,260		
Gold grade	oz/st	0.04			0.032	0.033	0.027	0.027	0.053	0.039	0.043	0.038	0.043		
Silver grade	oz/st	0.06			0.047	0.040	0.034	0.029	0.064	0.060	0.065	0.042	0.056		
Antimony grade	%	0.02%			0.02%	0.01%	0.02%	0.01%	0.02%	0.02%	0.02%	0.02%	0.02%		
Contained Gold	kozs	449			3.6	11.6	4.0	9.1	38.4	75.8	154.1	99.0	53.7		
Contained Silver	kozs	615			5.3	14.0	5.1	9.7	46.2	117.8	235.7	110.3	70.6		
Contained Antimony	klbs	4,319			48	84	72	40	332	746	1,595	998	403		
Gold Bullion Recovery	%	88.3%			89.1%	90.8%	89.8%	89.6%	86.3%	87.8%	88.1%	88.9%	88.8%		
Silver Bullion Recovery	%	5.1%			5.1%	5.1%	5.0%	5.3%	5.0%	5.1%	5.1%	5.1%	5.2%		
Recovered Gold	kozs	397			3	11	4	8	33	67	136	88	48		
Recovered Silver	kozs	31			0.27	0.72	0.26	0.51	2.33	6.01	12.02	5.58	3.64		

Stibnite Gold Project
Midas Gold Corporation

		Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Total Hangar Flats	kst	15,430			116	378	160	336	859	2,699	6,381	3,125	1,376		
Gold grade	oz/st	0.04			0.032	0.033	0.027	0.027	0.054	0.042	0.050	0.039	0.043		
Silver grade	oz/st	0.09			0.047	0.042	0.036	0.029	0.079	0.078	0.121	0.050	0.061		
Antimony grade	%	0.13%			0.02%	0.02%	0.03%	0.01%	0.09%	0.14%	0.22%	0.05%	0.04%		
Contained Gold	kozs	687			3.7	12.5	4.3	9.1	47	114	317	122	59		
Contained Silver	kozs	1,327			5.5	15.7	5.7	9.7	67	211	773	155	84		
Contained Antimony	klbs	40,757			55	155	94	40	1,587	7,399	27,492	2,938	997		
Gold Bullion Recovery	%	86.8%			89.1%	90.6%	89.5%	89.6%	85.8%	86.5%	85.8%	88.4%	88.7%		
Silver Bullion Recovery	%	5.1%			5.1%	5.1%	5.0%	5.3%	5.1%	5.1%	5.1%	5.1%	5.1%		
Recovered Gold	kozs	597			3.33	11.30	3.82	8.19	40.13	98.29	271.65	107.61	52.32		
Recovered Silver	kozs	68			0.28	0.80	0.28	0.51	3.42	10.76	39.42	7.83	4.34		
Antimony - Gold Recovery	%	0.8%			0.05%	0.07%	0.06%	-	0.39%	0.64%	1.24%	0.33%	0.16%		
Antimony - Silver Recovery	%	26.3%			1.5%	5.3%	5.1%	-	15.5%	21.6%	34.1%	14.0%	8.0%		
Antimony Recovery	%	73.7%			8.3%	33.6%	16.9%	-	65.6%	74.3%	78.1%	50.4%	46.8%		
Antimony Concentrate	kt	25.45			0.00	0.04	0.01		0.88	4.66	18.20	1.26	0.40		
Antimony Concentrate Grade	%	59%			59%	59%	59%		59%	59%	59%	59%	59%		
Recovered Gold	kozs	5			0.002	0.009	0.003		0.181	0.729	3.912	0.405	0.096		
Recovered Silver	kozs	349			0.08	0.83	0.29		10.43	45.56	263.65	21.70	6.76		
Recovered Antimony	klbs	30,030			4.6	52.0	15.9		1,041	5,496	21,472	1,481	466		
West End															
Low Antimony	kst	24,914							249	453	904	3,652	5,094	7,245	7,317
Gold grade	oz/st	0.041							0.038	0.057	0.048	0.041	0.037	0.041	0.043
Silver grade	oz/st	0.04							0.02	0.02	0.02	0.03	0.05	0.06	0.05
Contained Gold	kozs	1,024							10	26	43	149	186	299	312
Contained Silver	kozs	1,090							5	8	18	91	239	398	329
Gold Bullion Recovery	%	87.8%							88.6%	87.8%	87.8%	88.7%	89.0%	87.5%	87.0%
Silver Bullion Recovery	%	49.8%							49.5%	50.0%	48.9%	49.3%	50.1%	49.9%	49.7%
Recovered Gold	kozs	899							8	23	38	132	165	262	271
Recovered Silver	kozs	543							3	4	9	45	120	199	164
Oxide Feed	kst	10,736		1,911	660	660	660	658	674	681	765	1,273	1,580	805	409
Gold grade	oz/st	0.022		0.023	0.024	0.022	0.021	0.021	0.016	0.019	0.018	0.023	0.023	0.025	0.023
Silver grade	oz/st	0.03		0.03	0.02	0.03	0.03	0.02	0.01	0.02	0.02	0.03	0.04	0.05	0.05
Contained Gold	kozs	233		43.8	15.5	14.5	13.9	13.8	11.1	12.7	13.8	28.6	36.5	19.8	9.3
Contained Silver	kozs	314		47.8	16.2	17.2	17.2	13.2	9.4	15.7	15.3	39.5	63.2	41.1	18.4
Gold Bullion Recovery	%	81.9%		73.1%	69.9%	76.7%	74.6%	74.6%	86.5%	88.0%	86.6%	89.5%	88.9%	89.1%	87.0%
Silver Bullion Recovery	%	44.1%		44.1%	44.2%	44.8%	43.5%	43.8%	44.3%	44.3%	44.2%	43.6%	44.4%	44.0%	44.2%
Recovered Gold	kozs	191		32.0	10.8	11.1	10.3	10.3	9.6	11.1	12.0	25.6	32.4	17.6	8.1
Recovered Silver	kozs	138		21.1	7.2	7.7	7.5	5.8	4.2	6.9	6.8	17.2	28.1	18.1	8.1
Total West End	kst	35,650		1,911	660	660	660	658	923	1,134	1,669	4,925	6,674	8,050	7,726
Gold grade	oz/st	0.035		0.023	0.024	0.022	0.021	0.021	0.022	0.034	0.034	0.036	0.033	0.040	0.042
Silver grade	oz/st	0.04		0.03	0.02	0.03	0.03	0.02	0.02	0.02	0.02	0.03	0.05	0.05	0.05
Contained Gold	kozs	1,257		44	16	15	14	14	21	39	57	177	222	319	321
Contained Silver	kozs	1,404		48	16	17	17	13	15	23	33	131	303	440	348
Gold Bullion Recovery	%	86.7%		73.1%	69.9%	76.7%	74.6%	74.6%	87.5%	87.9%	87.5%	88.8%	89.0%	87.6%	87.0%
Silver Bullion Recovery	%	48.5%		44.1%	44.2%	44.8%	43.5%	43.8%	46.2%	46.2%	46.7%	47.6%	48.9%	49.4%	49.4%
Recovered Gold	kozs	1,090		32.0	10.8	11.1	10.3	10.3	18.0	34.0	49.7	157.4	197.9	279.5	279.4
Recovered Silver	kozs	681		21.1	7.2	7.7	7.5	5.8	6.9	10.8	15.6	62.2	148.0	217.0	171.7
Historic Tailings	kst	3,001		477	916	916	692								
Gold grade	oz/st	0.032		0.032	0.032	0.032	0.032								
Silver grade	oz/st	0.08		0.08	0.08	0.08	0.08								
Contained Gold	kozs	97		15	30	29	22								
Contained Silver	kozs	239		38	73	73	55								
Gold Bullion Recovery	%	74.5%		70.5%	69.7%	77.0%	80.3%								
Silver Bullion Recovery	%	8.5%		8.5%	8.5%	8.5%	8.5%								
Recovered Gold	kozs	72		11	21	23	18								
Recovered Silver	kozs	20		3.2	6.2	6.2	4.7								

Case A
Payables and Revenue

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,200/oz
Silver - $20.00/oz
Antimony - $4.00/lb

		Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Payable Metals															
Dore Metals															
Payable Gold - Dore	kozs	4,002		334	452	375	374	349	361	370	320	264	249	278	278
Payable Silver - Dore	kozs	1,085		89	79	53	83	46	43	39	54	69	149	213	168
Antimony Concentrate Payable Metals															
Antimony Concentrate	kst	84.6		12.5	11.5	5.7	17.9	5.6	6.1	5.3	18.2	1.3	0.4		
Payable Gold - Concentrate	kozs	3.2		0.43	0.49	0.43	0.50	0.15	0.16	0.22	0.67	0.08	0.02		
Payable Silver - Concentrate	kozs	382		54.3	46.8	53.7	68.0	18.3	11.6	23.5	94.6	8.8	2.8		
Payable Antimony - Concentrate	klbs	67,900		10,048	9,240	4,595	14,384	4,521	4,911	4,276	14,601	1,007	317		
Revenues								**Thounsands of US dollars ($000s)**							
Metal Prices															
Gold	$/oz	$1,200.00		$1,200.00	$1,200.00	$1,200.00	$1,200.00	$1,200.00	$1,200.00	$1,200.00	$1,200.00	$1,200.00	$1,200.00	$1,200.00	$1,200.00
Silver	$/oz	$20.00		$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00
Antimony	$/lb	$4.00		$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00
Dore															
Gold		$4,802,897		$400,573	$542,088	$449,482	$448,934	$418,817	$433,245	$443,603	$383,696	$316,433	$298,751	$333,669	$333,605
Silver		$21,697		$1,785	$1,590	$1,056	$1,655	$919	$856	$782	$1,078	$1,373	$2,986	$4,253	$3,365
Refining/Transport Cost															
Gold		$8,605		$718	$971	$805	$804	$750	$776	$795	$687	$567	$535	$598	$598
Silver		$1,790		$147	$131	$87	$136	$76	$71	$65	$89	$113	$246	$351	$278
Antimony Concentrate															
Gold		$3,782		$512	$589	$515	$600	$185	$189	$269	$806	$92	$26		
Silver		$7,645		$1,085	$935	$1,073	$1,359	$366	$233	$470	$1,891	$176	$57		
Antimony		$271,598		$40,190	$36,960	$18,380	$57,536	$18,083	$19,643	$17,103	$58,405	$4,028	$1,268		
Treatment/Transport Cost		$13,861		$2,051	$1,886	$938	$2,936	$923	$1,002	$873	$2,981	$206	$65		
Total Revenues		$5,083,363		$441,229	$579,173	$468,676	$506,206	$436,621	$452,316	$460,495	$442,120	$321,217	$302,242	$336,973	$336,095

Case A
Expenses and Results

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,200/oz
Silver - $20.00/oz
Antimony - $4.00/lb

	Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Operating Cost									**Thousands of US dollar ($000s)**							
Mining	$889,999		$1,999	$23,105	$62,065	$87,256	$92,092	$99,799	$88,174	$82,987	$69,083	$68,578	$70,381	$65,794	$47,885	$30,803
Process Plant	$1,416,809				$107,738	$124,488	$122,912	$124,017	$117,470	$117,015	$120,502	$116,896	$115,219	$115,447	$118,681	$116,426
G&A	$306,936				$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578
Total Operating Cost	$2,613,745		$1,999	$23,105	$195,380	$237,322	$240,581	$249,394	$231,222	$225,580	$215,162	$211,052	$211,178	$206,818	$192,145	$172,806
Royalty	$81,567				$6,806	$9,209	$7,636	$7,628	$7,110	$7,355	$7,532	$6,525	$5,371	$5,070	$5,662	$5,661
Property Taxes	$3,665				$315	$427	$389	$380	$317	$366	$346	$389	$137	$204	$167	$230
Salvage Value	-$26,524															
Reclamation/Closure																
Production Cost	$2,672,454		$1,999	$23,105	$202,502	$246,957	$248,607	$257,402	$238,650	$233,301	$223,040	$217,965	$216,686	$212,092	$197,973	$178,697
Net Operating Income	$2,410,909		-$1,999	-$23,105	$238,727	$332,216	$220,069	$248,804	$197,971	$219,015	$237,455	$224,154	$104,532	$90,149	$139,000	$157,398
Depreciation																
Initial Capital	$970,254			$171	$139,264	$237,132	$169,967	$121,993	$87,663	$85,587	$85,683	$42,793				
Sustaining Capital	$154,385				$299	$680	$2,314	$3,693	$4,434	$10,434	$14,129	$12,310	$11,494	$9,664	$8,123	$7,642
Total Depreciation	$1,124,639			$171	$139,563	$237,812	$172,281	$125,686	$92,098	$96,021	$99,812	$55,103	$11,494	$9,664	$8,123	$7,642
Net Income after Depreciation	$1,286,271		-$1,999	-$23,276	$99,164	$94,404	$47,788	$123,118	$105,873	$122,995	$137,643	$169,051	$93,037	$80,485	$130,877	$149,756
Idaho Mine License Tax	$7,259				$471	$462	$236	$616	$529	$615	$688	$981	$465	$402	$802	$992
Idaho Corporate Income Tax	$44,232							$3,717	$3,917	$4,551	$5,093	$7,258	$3,442	$2,978	$5,934	$7,342
Federal Income Tax	$193,725							$16,281	$17,157	$19,931	$22,305	$31,788	$15,077	$13,043	$25,989	$32,154
Net Income after Taxes	$1,041,055		-$1,999	-$23,276	$98,694	$93,942	$47,552	$102,503	$84,270	$97,898	$109,557	$129,024	$74,053	$64,062	$98,152	$109,268
Cash Flow																
Net Operating Income	$2,410,909		-$1,999	-$23,105	$238,727	$332,216	$220,069	$248,804	$197,971	$219,015	$237,455	$224,154	$104,532	$90,149	$139,000	$157,398
Working Capital																
Account Receivables					-$18,133	-$5,669	$4,541	-$1,542	$2,860	-$645	-$336	$755	$4,969	$780	-$1,427	$36
Accounts Payable					$8,029	$1,724	$134	$362	-$747	-$232	-$428	-$169	$5	-$179	-$603	-$795
Inventory (Parts)				-$7,500	-$7,500											
Total Working Capital	$0			-$7,500	-$17,603	-$3,945	$4,675	-$1,180	$2,113	-$877	-$764	$586	$4,974	$601	-$2,030	-$759
Capital Expenditures																
Initial Capital	$970,254	$137,371	$341,335	$491,548												
Sustaining Capital	$154,385				$2,094	$1,169	$11,627	$2,658	$9,916	$40,566	$5,589	$7,944	$9,039	$699	$1,797	$4,746
Total Capital Expenditures	$1,124,639	$137,371	$341,335	$491,548	$2,094	$1,169	$11,627	$2,658	$9,916	$40,566	$5,589	$7,944	$9,039	$699	$1,797	$4,746
Cash Flow before Taxes	$1,286,271	-$137,371	-$343,334	-$522,153	$219,030	$327,102	$213,117	$244,966	$190,168	$177,572	$231,101	$216,797	$100,467	$90,051	$135,172	$151,894
Cummulative Cash Flow before Taxes		-$137,371	-$480,705	-$1,002,858	-$783,829	-$456,727	-$243,609	$1,356	$191,524	$369,096	$600,197	$816,994	$917,461	$1,007,512	$1,142,684	$1,294,578
Taxes	$245,216				$471	$462	$236	$20,614	$21,603	$25,097	$28,086	$40,027	$18,984	$16,423	$32,725	$40,488
Cash Flow after Taxes	$1,041,055	-$137,371	-$343,334	-$522,153	$218,559	$326,640	$212,881	$224,351	$168,564	$152,475	$203,015	$176,770	$81,483	$73,628	$102,447	$111,406
Cummulative Cash Flow after Taxes		-$137,371	-$480,705	-$1,002,858	-$784,299	-$457,659	-$244,778	-$20,426	$148,138	$300,613	$503,628	$680,398	$761,881	$835,509	$937,956	$1,049,362

Economic Indicators before Taxes		
NPV @ 0%		$1,286,271
NPV @ 5%	5.0%	$662,398
NPV @ 7%	7.0%	$488,090
NPV @ 10%	10.0%	$281,855
IRR		16.2%
Payback	Years	4.0

Economic Indicators after Taxes		
NPV @ 0%		$1,041,055
NPV @ 5%	5.0%	$512,978
NPV @ 10%	10.0%	$187,239
IRR		14.4%
Payback	Years	4.1

Case A
Expenses and Results

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,200/oz
Silver - $20.00/oz
Antimony - $4.00/lb

	Total	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30
Operating Cost		**Thousands of US dollar ($000s)**																	
Mining	$889,999																		
Process Plant	$1,416,809																		
G&A	$306,936																		
Total Operating Cost	$2,613,745																		
Royalty	$81,567																		
Property Taxes	$3,665																		
Salvage Value	-$26,524		-$3,979	-$3,183	-$3,183	-$2,918	-$2,652	-$2,652	-$2,122	-$2,122	-$1,857	-$1,857							
Reclamation/Closure																			
Production Cost	$2,672,454		-$3,979	-$3,183	-$3,183	-$2,918	-$2,652	-$2,652	-$2,122	-$2,122	-$1,857	-$1,857							
Net Operating Income	$2,410,909		$3,979	$3,183	$3,183	$2,918	$2,652	$2,652	$2,122	$2,122	$1,857	$1,857							
Depreciation																			
Initial Capital	$970,254																		
Sustaining Capital	$154,385	$6,533	$6,419	$7,111	$6,278	$5,324	$4,955	$4,749	$4,388	$3,743	$3,351	$3,434	$3,263	$2,614	$1,937	$1,616	$1,393	$975	$1,085
Total Depreciation	$1,124,639	$6,533	$6,419	$7,111	$6,278	$5,324	$4,955	$4,749	$4,388	$3,743	$3,351	$3,434	$3,263	$2,614	$1,937	$1,616	$1,393	$975	$1,085
Net Income after Depreciation	$1,286,271	-$6,533	-$2,440	-$3,928	-$3,096	-$2,406	-$2,303	-$2,097	-$2,266	-$1,622	-$1,494	-$1,577	-$3,263	-$2,614	-$1,937	-$1,616	-$1,393	-$975	-$1,085
Idaho Mine License Tax	$7,259																		
Idaho Corporate Income Tax	$44,232																		
Federal Income Tax	$193,725																		
Net Income after Taxes	$1,041,055	-$6,533	-$2,440	-$3,928	-$3,096	-$2,406	-$2,303	-$2,097	-$2,266	-$1,622	-$1,494	-$1,577	-$3,263	-$2,614	-$1,937	-$1,616	-$1,393	-$975	-$1,085
Cash Flow																			
Net Operating Income	$2,410,909		$3,979	$3,183	$3,183	$2,918	$2,652	$2,652	$2,122	$2,122	$1,857	$1,857							
Working Capital																			
Account Receivables		$13,812																	
Accounts Payable		-$7,102																	
Inventory (Parts)				$15,000															
Total Working Capital	$0	$6,710		$15,000															
Capital Expenditures																			
Initial Capital	$970,254																		
Sustaining Capital	$154,385	$8,013	$11,019	$7,234	$3,703	$3,112	$3,073	$3,035	$2,806	$2,771	$4,177	$4,111	$1,716	$332	$249	$1,190			
Total Capital Expenditures	$1,124,639	$8,013	$11,019	$7,234	$3,703	$3,112	$3,073	$3,035	$2,806	$2,771	$4,177	$4,111	$1,716	$332	$249	$1,190			
Cash Flow before Taxes	$1,286,271	-$1,303	-$7,040	$10,949	-$520	-$194	-$421	-$382	-$685	-$649	-$2,320	-$2,255	-$1,716	-$332	-$249	-$1,190			
Cummulative Cash Flow before Taxes		$1,293,275	$1,286,235	$1,297,183	$1,296,664	$1,296,469	$1,296,048	$1,295,666	$1,294,981	$1,294,333	$1,292,013	$1,289,758	$1,288,042	$1,287,710	$1,287,461	$1,286,271	$1,286,271	$1,286,271	$1,286,271
Taxes	$245,216																		
Cash Flow after Taxes	$1,041,055	-$1,303	-$7,040	$10,949	-$520	-$194	-$421	-$382	-$685	-$649	-$2,320	-$2,255	-$1,716	-$332	-$249	-$1,190			
Cummulative Cash Flow after Taxes		$1,048,059	$1,041,019	$1,051,967	$1,051,448	$1,051,253	$1,050,832	$1,050,450	$1,049,765	$1,049,117	$1,046,797	$1,044,542	$1,042,826	$1,042,493	$1,042,245	$1,041,055	$1,041,055	$1,041,055	$1,041,055

Economic Indicators before Taxes		
NPV @ 0%		$1,286,271
NPV @ 5%	5.0%	$662,398
NPV @ 7%	7.0%	$488,090
NPV @ 10%	10.0%	$281,855
IRR		16.2%
Payback	Years	4.0

Economic Indicators after Taxes		
NPV @ 0%		$1,041,055
NPV @ 5%	5.0%	$512,978
NPV @ 10%	10.0%	$187,239
IRR		14.4%
Payback	Years	4.1

Case B (Base Case)
Payables and Revenue

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,350/oz
Silver - $22.50/oz
Antimony - $4.50/lb

		Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Payable Metals															
Dore Metals															
Payable Gold - Dore	kozs	4,002		334	452	375	374	349	361	370	320	264	249	278	278
Payable Silver - Dore	kozs	1,085		89	79	53	83	46	43	39	54	69	149	213	168
Antimony Concentrate Payable Metals															
Antimony Concentrate	kst	84.6		12.5	11.5	5.7	17.9	5.6	6.1	5.3	18.2	1.3	0.4		
Payable Gold - Concentrate	kozs	3.2		0.43	0.49	0.43	0.50	0.15	0.16	0.22	0.67	0.08	0.02		
Payable Silver - Concentrate	kozs	382		54.3	46.8	53.7	68.0	18.3	11.6	23.5	94.6	8.8	2.8		
Payable Antimony - Concentrate	klbs	67,900		10,048	9,240	4,595	14,384	4,521	4,911	4,276	14,601	1,007	317		
Revenues								**Thounsands of US dollars ($000s)**							
Metal Prices															
Gold	$/oz	$1,350.00		$1,350.00	$1,350.00	$1,350.00	$1,350.00	$1,350.00	$1,350.00	$1,350.00	$1,350.00	$1,350.00	$1,350.00	$1,350.00	$1,350.00
Silver	$/oz	$22.50		$22.50	$22.50	$22.50	$22.50	$22.50	$22.50	$22.50	$22.50	$22.50	$22.50	$22.50	$22.50
Antimony	$/lb	$4.50		$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50
Dore															
Gold		$5,403,259		$450,644	$609,849	$505,668	$505,050	$471,169	$487,401	$499,054	$431,658	$355,987	$336,095	$375,378	$375,306
Silver		$24,409		$2,008	$1,788	$1,188	$1,861	$1,034	$963	$880	$1,213	$1,545	$3,359	$4,784	$3,786
Refining/Transport Cost															
Gold		$8,605		$718	$971	$805	$804	$750	$776	$795	$687	$567	$535	$598	$598
Silver		$1,790		$147	$131	$87	$136	$76	$71	$65	$89	$113	$246	$351	$278
Antimony Concentrate															
Gold		$4,255		$576	$662	$579	$675	$208	$212	$303	$907	$104	$29		
Silver		$8,601		$1,221	$1,052	$1,208	$1,529	$411	$262	$529	$2,128	$198	$64		
Antimony		$305,548		$45,214	$41,580	$20,677	$64,728	$20,344	$22,098	$19,241	$65,706	$4,532	$1,427		
Treatment/Transport Cost		$13,861		$2,051	$1,886	$938	$2,936	$923	$1,002	$873	$2,981	$206	$65		
Total Revenues		$5,721,816		$496,747	$651,944	$527,489	$569,966	$491,417	$509,087	$518,273	$497,854	$361,480	$340,128	$379,213	$378,216

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,350/oz
Silver - $22.50/oz
Antimony - $4.50/lb

	Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Operating Cost								**Thousands of US dollar ($000s)**								
Mining	$889,999		$1,999	$23,105	$62,065	$87,256	$92,092	$99,799	$88,174	$82,987	$69,083	$68,578	$70,381	$65,794	$47,885	$30,803
Process Plant	$1,416,809				$107,738	$124,488	$122,912	$124,017	$117,470	$117,015	$120,502	$116,896	$115,219	$115,447	$118,681	$116,426
G&A	$306,936				$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578
Total Operating Cost	$2,613,745		$1,999	$23,105	$195,380	$237,322	$240,581	$249,394	$231,222	$225,580	$215,162	$211,052	$211,178	$206,818	$192,145	$172,806
Royalty	$91,781				$7,659	$10,362	$8,593	$8,584	$8,001	$8,276	$8,476	$7,342	$6,044	$5,705	$6,371	$6,370
Property Taxes	$3,665				$315	$427	$389	$380	$317	$366	$346	$389	$137	$204	$167	$230
Salvage Value	-$26,524															
Reclamation/Closure																
Production Cost	$2,682,668		$1,999	$23,105	$203,354	$248,110	$249,563	$258,358	$239,540	$234,222	$223,983	$218,782	$217,358	$212,727	$198,683	$179,406
Net Operating Income	$3,039,148		-$1,999	-$23,105	$293,393	$403,833	$277,926	$311,609	$251,877	$274,865	$294,290	$279,072	$144,122	$127,400	$180,531	$198,810
Depreciation																
Initial Capital	$970,254			$171	$139,264	$237,132	$169,967	$121,993	$87,663	$85,587	$85,683	$42,793				
Sustaining Capital	$154,385				$299	$680	$2,314	$3,693	$4,434	$10,434	$14,129	$12,310	$11,494	$9,664	$8,123	$7,642
Total Depreciation	$1,124,639			$171	$139,563	$237,812	$172,281	$125,686	$92,098	$96,021	$99,812	$55,103	$11,494	$9,664	$8,123	$7,642
Net Income after Depreciation	$1,914,509		-$1,999	-$23,276	$153,830	$166,021	$105,645	$185,923	$159,779	$178,844	$194,478	$223,968	$132,628	$117,736	$172,408	$191,168
Idaho Mine License Tax	$11,426				$744	$820	$525	$953	$844	$1,007	$1,151	$1,441	$780	$665	$1,154	$1,343
Idaho Corporate Income Tax	$75,072					$2,089	$3,887	$7,053	$6,244	$7,449	$8,520	$10,665	$5,769	$4,920	$8,538	$9,939
Federal Income Tax	$328,797					$9,147	$17,023	$30,892	$27,348	$32,623	$37,314	$46,712	$25,265	$21,550	$37,396	$43,528
Net Income after Taxes	$1,499,214		-$1,999	-$23,276	$153,086	$153,965	$84,210	$147,025	$125,344	$137,766	$147,493	$165,150	$100,814	$90,602	$125,320	$136,359
Cash Flow																
Net Operating Income	$3,039,148		-$1,999	-$23,105	$293,393	$403,833	$277,926	$311,609	$251,877	$274,865	$294,290	$279,072	$144,122	$127,400	$180,531	$198,810
Working Capital																
Account Receivables					-$20,414	-$6,378	$5,115	-$1,746	$3,228	-$726	-$378	$839	$5,604	$878	-$1,606	$41
Accounts Payable					$8,029	$1,724	$134	$362	-$747	-$232	-$428	-$169	$5	-$179	-$603	-$795
Inventory (Parts)				-$7,500	-$7,500											
Total Working Capital	$0			-$7,500	-$19,885	-$4,654	$5,249	-$1,383	$2,481	-$958	-$806	$670	$5,610	$698	-$2,209	-$754
Capital Expenditures																
Initial Capital	$970,254	$137,371	$341,335	$491,548												
Sustaining Capital	$154,385				$2,094	$1,169	$11,627	$2,658	$9,916	$40,566	$5,589	$7,944	$9,039	$699	$1,797	$4,746
Total Capital Expenditures	$1,124,639	$137,371	$341,335	$491,548	$2,094	$1,169	$11,627	$2,658	$9,916	$40,566	$5,589	$7,944	$9,039	$699	$1,797	$4,746
Cash Flow before Taxes	$1,914,509	-$137,371	-$343,334	-$522,153	$271,414	$398,010	$271,548	$307,567	$244,442	$233,340	$287,895	$271,798	$140,693	$127,400	$176,524	$193,311
Cummulative Cash Flow before Taxes		-$137,371	-$480,705	-$1,002,858	-$731,444	-$333,434	-$61,886	$245,681	$490,123	$723,464	$1,011,359	$1,283,157	$1,423,850	$1,551,250	$1,727,774	$1,921,085
Taxes	$415,295				$744	$12,056	$21,434	$38,898	$34,436	$41,078	$46,985	$58,818	$31,814	$27,135	$47,088	$54,810
Cash Flow after Taxes	$1,499,214	-$137,371	-$343,334	-$522,153	$270,670	$385,954	$250,114	$268,669	$210,007	$192,262	$240,910	$212,980	$108,879	$100,265	$129,436	$138,501
Cummulative Cash Flow after Taxes		-$137,371	-$480,705	-$1,002,858	-$732,188	-$346,234	-$96,121	$172,549	$382,555	$574,818	$815,728	$1,028,708	$1,137,587	$1,237,852	$1,367,288	$1,505,790

Economic Indicators before Taxes		
NPV @ 0%		$1,914,509
NPV @ 5%	5.0%	$1,093,174
NPV @ 7%	7.0%	$862,974
NPV @ 10%	10.0%	$589,719
IRR		22.0%
Payback	Years	3.2

Economic Indicators after Taxes		
NPV @ 0%		$1,499,214
NPV @ 5%	5.0%	$831,755
NPV @ 10%	10.0%	$418,353
IRR		19.3%
Payback	Years	3.4

Case B (Base Case)
Expenses and Results

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,350/oz
Silver - $22.50/oz
Antimony - $4.50/lb

	Total	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30
Operating Cost		**Thousands of US dollar ($000s)**																	
Mining	$889,999																		
Process Plant	$1,416,809																		
G&A	$306,936																		
Total Operating Cost	$2,613,745																		
Royalty	$91,781																		
Property Taxes	$3,665																		
Salvage Value	-$26,524		-$3,979	-$3,183	-$3,183	-$2,918	-$2,652	-$2,652	-$2,122	-$2,122	-$1,857	-$1,857							
Reclamation/Closure																			
Production Cost	$2,682,668		-$3,979	-$3,183	-$3,183	-$2,918	-$2,652	-$2,652	-$2,122	-$2,122	-$1,857	-$1,857							
Net Operating Income	$3,039,148		$3,979	$3,183	$3,183	$2,918	$2,652	$2,652	$2,122	$2,122	$1,857	$1,857							
Depreciation																			
Initial Capital	$970,254																		
Sustaining Capital	$154,385	$6,533	$6,419	$7,111	$6,278	$5,324	$4,955	$4,749	$4,388	$3,743	$3,351	$3,434	$3,263	$2,614	$1,937	$1,616	$1,393	$975	$1,085
Total Depreciation	$1,124,639	$6,533	$6,419	$7,111	$6,278	$5,324	$4,955	$4,749	$4,388	$3,743	$3,351	$3,434	$3,263	$2,614	$1,937	$1,616	$1,393	$975	$1,085
Net Income after Depreciation	$1,914,509	-$6,533	-$2,440	-$3,928	-$3,096	-$2,406	-$2,303	-$2,097	-$2,266	-$1,622	-$1,494	-$1,577	-$3,263	-$2,614	-$1,937	-$1,616	-$1,393	-$975	-$1,085
Idaho Mine License Tax	$11,426																		
Idaho Corporate Income Tax	$75,072																		
Federal Income Tax	$328,797																		
Net Income after Taxes	$1,499,214	-$6,533	-$2,440	-$3,928	-$3,096	-$2,406	-$2,303	-$2,097	-$2,266	-$1,622	-$1,494	-$1,577	-$3,263	-$2,614	-$1,937	-$1,616	-$1,393	-$975	-$1,085
Cash Flow																			
Net Operating Income	$3,039,148		$3,979	$3,183	$3,183	$2,918	$2,652	$2,652	$2,122	$2,122	$1,857	$1,857							
Working Capital																			
Account Receivables		$15,543																	
Accounts Payable		-$7,102																	
Inventory (Parts)				$15,000															
Total Working Capital	$0	$8,442		$15,000															
Capital Expenditures																			
Initial Capital	$970,254																		
Sustaining Capital	$154,385	$8,013	$11,019	$7,234	$3,703	$3,112	$3,073	$3,035	$2,806	$2,771	$4,177	$4,111	$1,716	$332	$249	$1,190			
Total Capital Expenditures	$1,124,639	$8,013	$11,019	$7,234	$3,703	$3,112	$3,073	$3,035	$2,806	$2,771	$4,177	$4,111	$1,716	$332	$249	$1,190			
Cash Flow before Taxes	$1,914,509	$428	-$7,040	$10,949	-$520	-$194	-$421	-$382	-$685	-$649	-$2,320	-$2,255	-$1,716	-$332	-$249	-$1,190			
Cummulative Cash Flow before Taxes		$1,921,513	$1,914,473	$1,925,422	$1,924,902	$1,924,708	$1,924,287	$1,923,904	$1,923,220	$1,922,571	$1,920,251	$1,917,996	$1,916,280	$1,915,948	$1,915,699	$1,914,509	$1,914,509	$1,914,509	$1,914,509
Taxes	$415,295																		
Cash Flow after Taxes	$1,499,214	$428	-$7,040	$10,949	-$520	-$194	-$421	-$382	-$685	-$649	-$2,320	-$2,255	-$1,716	-$332	-$249	-$1,190			
Cummulative Cash Flow after Taxes		$1,506,218	$1,499,178	$1,510,126	$1,509,607	$1,509,412	$1,508,991	$1,508,609	$1,507,924	$1,507,276	$1,504,956	$1,502,701	$1,500,985	$1,500,652	$1,500,404	$1,499,214	$1,499,214	$1,499,214	$1,499,214

Economic Indicators before Taxes		
NPV @ 0%		$1,914,509
NPV @ 5%	5.0%	$1,093,174
NPV @ 7%	7.0%	$862,974
NPV @ 10%	10.0%	$589,719
IRR		22.0%
Payback	Years	3.2

Economic Indicators after Taxes		
NPV @ 0%		$1,499,214
NPV @ 5%	5.0%	$831,755
NPV @ 10%	10.0%	$418,353
IRR		19.3%
Payback	Years	3.4

Case C
Payables and Revenue

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,500/oz
Silver - $25.00/oz
Antimony - $5.00/lb

		Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Payable Metals															
Dore Metals															
Payable Gold - Dore	kozs	4,002		334	452	375	374	349	361	370	320	264	249	278	278
Payable Silver - Dore	kozs	1,085		89	79	53	83	46	43	39	54	69	149	213	168
Antimony Concentrate Payable Metals															
Antimony Concentrate	kst	84.6		12.5	11.5	5.7	17.9	5.6	6.1	5.3	18.2	1.3	0.4		
Payable Gold - Concentrate	kozs	3.2		0.43	0.49	0.43	0.50	0.15	0.16	0.22	0.67	0.08	0.02		
Payable Silver - Concentrate	kozs	382		54.3	46.8	53.7	68.0	18.3	11.6	23.5	94.6	8.8	2.8		
Payable Antimony - Concentrate	klbs	67,900		10,048	9,240	4,595	14,384	4,521	4,911	4,276	14,601	1,007	317		
Revenues								**Thounsands of US dollars ($000s)**							
Metal Prices															
Gold	$/oz	$1,500.00		$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00
Silver	$/oz	$25.00		$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00
Antimony	$/lb	$5.00		$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00
Dore															
Gold		$6,003,621		$500,716	$677,610	$561,853	$561,167	$523,521	$541,557	$554,504	$479,620	$395,542	$373,439	$417,086	$417,006
Silver		$27,121		$2,231	$1,987	$1,320	$2,068	$1,149	$1,070	$977	$1,348	$1,716	$3,733	$5,316	$4,207
Refining/Transport Cost															
Gold		$8,605		$718	$971	$805	$804	$750	$776	$795	$687	$567	$535	$598	$598
Silver		$1,790		$147	$131	$87	$136	$76	$71	$65	$89	$113	$246	$351	$278
Antimony Concentrate															
Gold		$4,728		$640	$736	$644	$750	$231	$236	$336	$1,008	$116	$32		
Silver		$9,556		$1,356	$1,169	$1,342	$1,699	$457	$291	$587	$2,364	$221	$71		
Antimony		$339,498		$50,238	$46,200	$22,975	$71,920	$22,604	$24,554	$21,379	$73,006	$5,035	$1,585		
Treatment/Transport Cost		$13,861		$2,051	$1,886	$938	$2,936	$923	$1,002	$873	$2,981	$206	$65		
Total Revenues		$6,360,268		$552,265	$724,714	$586,302	$633,727	$546,213	$565,858	$576,052	$553,589	$401,743	$378,014	$421,454	$420,338

Case C
Expenses and Results

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,500/oz
Silver - $25.00/oz
Antimony - $5.00/lb

	Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Operating Cost									**Thousands of US dollar ($000s)**							
Mining	$889,999		$1,999	$23,105	$62,065	$87,256	$92,092	$99,799	$88,174	$82,987	$69,083	$68,578	$70,381	$65,794	$47,885	$30,803
Process Plant	$1,416,809				$107,738	$124,488	$122,912	$124,017	$117,470	$117,015	$120,502	$116,896	$115,219	$115,447	$118,681	$116,426
G&A	$306,936				$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578
Total Operating Cost	$2,613,745		$1,999	$23,105	$195,380	$237,322	$240,581	$249,394	$231,222	$225,580	$215,162	$211,052	$211,178	$206,818	$192,145	$172,806
Royalty	$101,996				$8,511	$11,515	$9,549	$9,539	$8,891	$9,197	$9,419	$8,159	$6,717	$6,340	$7,080	$7,079
Property Taxes	$3,665				$315	$427	$389	$380	$317	$366	$346	$389	$137	$204	$167	$230
Salvage Value	-$26,524															
Reclamation/Closure																
Production Cost	$2,692,882		$1,999	$23,105	$204,206	$249,264	$250,519	$259,313	$240,431	$235,143	$224,927	$219,600	$218,031	$213,362	$199,392	$180,115
Net Operating Income	$3,667,386		-$1,999	-$23,105	$348,059	$475,450	$335,783	$374,414	$305,783	$330,715	$351,125	$333,989	$183,712	$164,651	$222,062	$240,223
Depreciation																
Initial Capital	$970,254			$171	$139,264	$237,132	$169,967	$121,993	$87,663	$85,587	$85,683	$42,793				
Sustaining Capital	$154,385				$299	$680	$2,314	$3,693	$4,434	$10,434	$14,129	$12,310	$11,494	$9,664	$8,123	$7,642
Total Depreciation	$1,124,639			$171	$139,563	$237,812	$172,281	$125,686	$92,098	$96,021	$99,812	$55,103	$11,494	$9,664	$8,123	$7,642
Net Income after Depreciation	$2,542,747		-$1,999	-$23,276	$208,496	$237,638	$163,502	$248,728	$213,685	$234,694	$251,313	$278,886	$172,218	$154,988	$213,939	$232,581
Idaho Mine License Tax	$16,299				$1,167	$1,232	$815	$1,481	$1,299	$1,478	$1,631	$1,902	$1,115	$980	$1,506	$1,694
Idaho Corporate Income Tax	$111,130					$8,266	$6,027	$10,956	$9,613	$10,939	$12,073	$14,073	$8,249	$7,256	$11,143	$12,535
Federal Income Tax	$486,719					$36,204	$26,398	$47,984	$42,103	$47,908	$52,877	$61,635	$36,128	$31,777	$48,803	$54,902
Net Income after Taxes	$1,928,599		-$1,999	-$23,276	$207,329	$191,937	$130,262	$188,308	$160,669	$174,369	$184,732	$201,276	$126,727	$114,974	$152,488	$163,449
Cash Flow																
Net Operating Income	$3,667,386		-$1,999	-$23,105	$348,059	$475,450	$335,783	$374,414	$305,783	$330,715	$351,125	$333,989	$183,712	$164,651	$222,062	$240,223
Working Capital																
Account Receivables					-$22,696	-$7,087	$5,688	-$1,949	$3,596	-$807	-$419	$923	$6,240	$975	-$1,785	$46
Accounts Payable					$8,029	$1,724	$134	$362	-$747	-$232	-$428	-$169	$5	-$179	-$603	-$795
Inventory (Parts)				-$7,500	-$7,500											
Total Working Capital	$0			-$7,500	-$22,167	-$5,363	$5,822	-$1,587	$2,850	-$1,039	-$847	$754	$6,245	$796	-$2,388	-$749
Capital Expenditures																
Initial Capital	$970,254	$137,371	$341,335	$491,548												
Sustaining Capital	$154,385				$2,094	$1,169	$11,627	$2,658	$9,916	$40,566	$5,589	$7,944	$9,039	$699	$1,797	$4,746
Total Capital Expenditures	$1,124,639	$137,371	$341,335	$491,548	$2,094	$1,169	$11,627	$2,658	$9,916	$40,566	$5,589	$7,944	$9,039	$699	$1,797	$4,746
Cash Flow before Taxes	$2,542,747	-$137,371	-$343,334	-$522,153	$323,798	$468,918	$329,979	$370,169	$298,716	$289,109	$344,689	$326,800	$180,919	$164,749	$217,876	$234,728
Cummulative Cash Flow before Taxes		-$137,371	-$480,705	-$1,002,858	-$679,060	-$210,142	$119,837	$490,006	$788,722	########	$1,422,520	$1,749,320	$1,930,239	$2,094,988	$2,312,864	$2,547,592
Taxes	$614,148				$1,167	$45,702	$33,240	$60,420	$53,016	$60,325	$66,581	$77,610	$45,491	$40,013	$61,451	$69,132
Cash Flow after Taxes	$1,928,599	-$137,371	-$343,334	-$522,153	$322,631	$423,216	$296,738	$309,749	$245,701	$228,784	$278,108	$249,190	$135,428	$124,735	$156,425	$165,597
Cummulative Cash Flow after Taxes		-$137,371	-$480,705	-$1,002,858	-$680,227	-$257,010	$39,728	$349,477	$595,178	$823,962	$1,102,069	$1,351,259	$1,486,687	$1,611,423	$1,767,848	$1,933,444

Economic Indicators before Taxes		
NPV @ 0%		$2,542,747
NPV @ 5%	5.0%	$1,523,949
NPV @ 7%	7.0%	$1,237,858
NPV @ 10%	10.0%	$897,583
IRR		27.2%
Payback	Years	2.6

Economic Indicators after Taxes		
NPV @ 0%		$1,928,599
NPV @ 5%	5.0%	$1,128,756
NPV @ 10%	10.0%	$632,542
IRR		23.4%
Payback	Years	2.9

Case C
Expenses and Results

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,500/oz
Silver - $25.00/oz
Antimony - $5.00/lb

	Total	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30
Operating Cost		Thousands of US dollar ($000s)																	
Mining	$889,999																		
Process Plant	$1,416,809																		
G&A	$306,936																		
Total Operating Cost	$2,613,745																		
Royalty	$101,996																		
Property Taxes	$3,665																		
Salvage Value	-$26,524		-$3,979	-$3,183	-$3,183	-$2,918	-$2,652	-$2,652	-$2,122	-$2,122	-$1,857	-$1,857							
Reclamation/Closure																			
Production Cost	$2,692,882		-$3,979	-$3,183	-$3,183	-$2,918	-$2,652	-$2,652	-$2,122	-$2,122	-$1,857	-$1,857							
Net Operating Income	$3,667,386		$3,979	$3,183	$3,183	$2,918	$2,652	$2,652	$2,122	$2,122	$1,857	$1,857							
Depreciation																			
Initial Capital	$970,254																		
Sustaining Capital	$154,385	$6,533	$6,419	$7,111	$6,278	$5,324	$4,955	$4,749	$4,388	$3,743	$3,351	$3,434	$3,263	$2,614	$1,937	$1,616	$1,393	$975	$1,085
Total Depreciation	$1,124,639	$6,533	$6,419	$7,111	$6,278	$5,324	$4,955	$4,749	$4,388	$3,743	$3,351	$3,434	$3,263	$2,614	$1,937	$1,616	$1,393	$975	$1,085
Net Income after Depreciation	$2,542,747	-$6,533	-$2,440	-$3,928	-$3,096	-$2,406	-$2,303	-$2,097	-$2,266	-$1,622	-$1,494	-$1,577	-$3,263	-$2,614	-$1,937	-$1,616	-$1,393	-$975	-$1,085
Idaho Mine License Tax	$16,299																		
Idaho Corporate Income Tax	$111,130																		
Federal Income Tax	$486,719																		
Net Income after Taxes	$1,928,599	-$6,533	-$2,440	-$3,928	-$3,096	-$2,406	-$2,303	-$2,097	-$2,266	-$1,622	-$1,494	-$1,577	-$3,263	-$2,614	-$1,937	-$1,616	-$1,393	-$975	-$1,085
Cash Flow																			
Net Operating Income	$3,667,386		$3,979	$3,183	$3,183	$2,918	$2,652	$2,652	$2,122	$2,122	$1,857	$1,857							
Working Capital																			
Account Receivables		$17,274																	
Accounts Payable		-$7,102																	
Inventory (Parts)				$15,000															
Total Working Capital	$0	$10,173		$15,000															
Capital Expenditures																			
Initial Capital	$970,254																		
Sustaining Capital	$154,385	$8,013	$11,019	$7,234	$3,703	$3,112	$3,073	$3,035	$2,806	$2,771	$4,177	$4,111	$1,716	$332	$249	$1,190			
Total Capital Expenditures	$1,124,639	$8,013	$11,019	$7,234	$3,703	$3,112	$3,073	$3,035	$2,806	$2,771	$4,177	$4,111	$1,716	$332	$249	$1,190			
Cash Flow before Taxes	$2,542,747	$2,159	-$7,040	$10,949	-$520	-$194	-$421	-$382	-$685	-$649	-$2,320	-$2,255	-$1,716	-$332	-$249	-$1,190			
Cummulative Cash Flow before Taxes		$2,549,751	$2,542,711	$2,553,660	$2,553,140	$2,552,946	$2,552,525	$2,552,142	$2,551,458	$2,550,809	$2,548,489	$2,546,234	$2,544,518	$2,544,186	$2,543,937	$2,542,747	$2,542,747	$2,542,747	$2,542,747
Taxes	$614,148																		
Cash Flow after Taxes	$1,928,599	$2,159	-$7,040	$10,949	-$520	-$194	-$421	-$382	-$685	-$649	-$2,320	-$2,255	-$1,716	-$332	-$249	-$1,190			
Cummulative Cash Flow after Taxes		$1,935,604	$1,928,563	$1,939,512	$1,938,992	$1,938,798	$1,938,377	$1,937,994	$1,937,310	$1,936,661	$1,934,341	$1,932,086	$1,930,370	$1,930,038	$1,929,789	$1,928,599	$1,928,599	$1,928,599	$1,928,599

Economic Indicators before Taxes		
NPV @ 0%		$2,542,747
NPV @ 5%	5.0%	$1,523,949
NPV @ 7%	7.0%	$1,237,858
NPV @ 10%	10.0%	$897,583
IRR		27.2%
Payback	Years	2.6

Economic Indicators after Taxes		
NPV @ 0%		$1,928,599
NPV @ 5%	5.0%	$1,128,756
NPV @ 10%	10.0%	$632,542
IRR		23.4%
Payback	Years	2.9

Case D
Payables and Revenue

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,650/oz
Silver - $27.50/oz
Antimony - $5.50/lb

		Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Payable Metals															
Dore Metals															
Payable Gold - Dore	kozs	4,002		334	452	375	374	349	361	370	320	264	249	278	278
Payable Silver - Dore	kozs	1,085		89	79	53	83	46	43	39	54	69	149	213	168
Antimony Concentrate Payable Metals															
Antimony Concentrate	kst	84.6		12.5	11.5	5.7	17.9	5.6	6.1	5.3	18.2	1.3	0.4		
Payable Gold - Concentrate	kozs	3.2		0.43	0.49	0.43	0.50	0.15	0.16	0.22	0.67	0.08	0.02		
Payable Silver - Concentrate	kozs	382		54.3	46.8	53.7	68.0	18.3	11.6	23.5	94.6	8.8	2.8		
Payable Antimony - Concentrate	klbs	67,900		10,048	9,240	4,595	14,384	4,521	4,911	4,276	14,601	1,007	317		
Revenues							**Thounsands of US dollars ($000s)**								
Metal Prices															
Gold	$/oz	$1,650.00		$1,650.00	$1,650.00	$1,650.00	$1,650.00	$1,650.00	$1,650.00	$1,650.00	$1,650.00	$1,650.00	$1,650.00	$1,650.00	$1,650.00
Silver	$/oz	$27.50		$27.50	$27.50	$27.50	$27.50	$27.50	$27.50	$27.50	$27.50	$27.50	$27.50	$27.50	$27.50
Antimony	$/lb	$5.50		$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50
Dore															
Gold		$6,603,983		$550,788	$745,371	$618,038	$617,284	$575,873	$595,713	$609,954	$527,582	$435,096	$410,783	$458,795	$458,707
Silver		$29,834		$2,454	$2,186	$1,452	$2,275	$1,264	$1,177	$1,075	$1,483	$1,888	$4,106	$5,847	$4,627
Refining/Transport Cost															
Gold		$8,605		$718	$971	$805	$804	$750	$776	$795	$687	$567	$535	$598	$598
Silver		$1,790		$147	$131	$87	$136	$76	$71	$65	$89	$113	$246	$351	$278
Antimony Concentrate															
Gold		$5,200		$704	$809	$708	$824	$254	$259	$370	$1,108	$127	$35		
Silver		$10,512		$1,492	$1,286	$1,476	$1,869	$503	$320	$646	$2,600	$243	$78		
Antimony		$373,447		$55,262	$50,820	$25,272	$79,112	$24,864	$27,009	$23,517	$80,307	$5,539	$1,744		
Treatment/Transport Cost		$13,861		$2,051	$1,886	$938	$2,936	$923	$1,002	$873	$2,981	$206	$65		
Total Revenues		$6,998,720		$607,783	$797,484	$645,116	$697,487	$601,010	$622,628	$633,830	$609,323	$442,006	$415,900	$463,694	$462,459

Case D
Expenses and Results

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,650/oz
Silver - $27.50/oz
Antimony - $5.50/lb

	Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Operating Cost								**Thousands of US dollar ($000s)**								
Mining	$889,999		$1,999	$23,105	$62,065	$87,256	$92,092	$99,799	$88,174	$82,987	$69,083	$68,578	$70,381	$65,794	$47,885	$30,803
Process Plant	$1,416,809				$107,738	$124,488	$122,912	$124,017	$117,470	$117,015	$120,502	$116,896	$115,219	$115,447	$118,681	$116,426
G&A	$306,936				$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578	$25,578
Total Operating Cost	$2,613,745		$1,999	$23,105	$195,380	$237,322	$240,581	$249,394	$231,222	$225,580	$215,162	$211,052	$211,178	$206,818	$192,145	$172,806
Royalty	$112,210				$9,363	$12,669	$10,505	$10,494	$9,781	$10,118	$10,362	$8,976	$7,389	$6,975	$7,789	$7,788
Property Taxes	$3,665				$315	$427	$389	$380	$317	$366	$346	$389	$137	$204	$167	$230
Salvage Value	-$26,524															
Reclamation/Closure																
Production Cost	$2,703,096		$1,999	$23,105	$205,059	$250,417	$251,475	$260,268	$241,321	$236,064	$225,870	$220,417	$218,704	$213,997	$200,101	$180,824
Net Operating Income	$4,295,624		-$1,999	-$23,105	$402,725	$547,067	$393,640	$437,219	$359,689	$386,564	$407,960	$388,907	$223,303	$201,903	$263,593	$281,635
Depreciation																
Initial Capital	$970,254			$171	$139,264	$237,132	$169,967	$121,993	$87,663	$85,587	$85,683	$42,793				
Sustaining Capital	$154,385				$299	$680	$2,314	$3,693	$4,434	$10,434	$14,129	$12,310	$11,494	$9,664	$8,123	$7,642
Total Depreciation	$1,124,639			$171	$139,563	$237,812	$172,281	$125,686	$92,098	$96,021	$99,812	$55,103	$11,494	$9,664	$8,123	$7,642
Net Income after Depreciation	$3,170,985		-$1,999	-$23,276	$263,162	$309,255	$221,359	$311,533	$267,591	$290,544	$308,148	$333,803	$211,808	$192,239	$255,470	$273,993
Idaho Mine License Tax	$21,516				$1,627	$1,836	$1,219	$2,008	$1,754	$1,950	$2,112	$2,362	$1,450	$1,296	$1,858	$2,045
Idaho Corporate Income Tax	$149,737				$2,554	$13,583	$9,024	$14,859	$12,982	$14,429	$15,626	$17,480	$10,729	$9,591	$13,747	$15,132
Federal Income Tax	$655,807				$11,185	$59,491	$39,523	$65,076	$56,859	$63,193	$68,439	$76,559	$46,990	$42,005	$60,209	$66,276
Net Income after Taxes	$2,343,925		-$1,999	-$23,276	$247,796	$234,345	$171,592	$229,590	$195,995	$210,972	$221,971	$237,402	$152,639	$139,347	$179,656	$190,539
Cash Flow																
Net Operating Income	$4,295,624		-$1,999	-$23,105	$402,725	$547,067	$393,640	$437,219	$359,689	$386,564	$407,960	$388,907	$223,303	$201,903	$263,593	$281,635
Working Capital																
Account Receivables					-$24,977	-$7,796	$6,262	-$2,152	$3,965	-$888	-$460	$1,007	$6,876	$1,073	-$1,964	$51
Accounts Payable					$8,029	$1,724	$134	$362	-$747	-$232	-$428	-$169	$5	-$179	-$603	-$795
Inventory (Parts)				-$7,500	-$7,500											
Total Working Capital	$0			-$7,500	-$24,448	-$6,072	$6,396	-$1,790	$3,218	-$1,120	-$888	$838	$6,881	$894	-$2,567	-$744
Capital Expenditures																
Initial Capital	$970,254	$137,371	$341,335	$491,548												
Sustaining Capital	$154,385				$2,094	$1,169	$11,627	$2,658	$9,916	$40,566	$5,589	$7,944	$9,039	$699	$1,797	$4,746
Total Capital Expenditures	$1,124,639	$137,371	$341,335	$491,548	$2,094	$1,169	$11,627	$2,658	$9,916	$40,566	$5,589	$7,944	$9,039	$699	$1,797	$4,746
Cash Flow before Taxes	$3,170,985	-$137,371	-$343,334	-$522,153	$376,182	$539,826	$388,409	$432,771	$352,991	$344,877	$401,483	$381,801	$221,145	$202,097	$259,229	$276,145
Cummulative Cash Flow before Taxes		-$137,371	-$480,705	-$1,002,858	-$626,676	-$86,850	$301,559	$734,331	$1,087,321	$1,432,199	$1,833,681	$2,215,483	$2,436,628	$2,638,725	$2,897,954	$3,174,099
Taxes	$827,060				$15,366	$74,910	$49,767	$81,943	$71,596	$79,572	$86,177	$96,401	$59,169	$52,892	$75,814	$83,454
Cash Flow after Taxes	$2,343,925	-$137,371	-$343,334	-$522,153	$360,817	$464,916	$338,642	$350,828	$281,395	$265,306	$315,305	$285,400	$161,976	$149,206	$183,414	$192,692
Cummulative Cash Flow after Taxes		-$137,371	-$480,705	-$1,002,858	-$642,041	-$177,125	$161,517	$512,345	$793,740	$1,059,046	$1,374,351	$1,659,751	$1,821,727	$1,970,933	$2,154,347	$2,347,039

Economic Indicators before Taxes		
NPV @ 0%		$3,170,985
NPV @ 5%	5.0%	$1,954,725
NPV @ 7%	7.0%	$1,612,742
NPV @ 10%	10.0%	$1,205,447
IRR		31.9%
Payback	Years	2.2

Economic Indicators after Taxes		
NPV @ 0%		$2,343,925
NPV @ 5%	5.0%	$1,413,808
NPV @ 10%	10.0%	$836,479
IRR		27.0%
Payback	Years	2.5

Case D
Expenses and Results

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,650/oz
Silver - $27.50/oz
Antimony - $5.50/lb

	Total	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30
Operating Cost		**Thousands of US dollar ($000s)**																	
Mining	$889,999																		
Process Plant	$1,416,809																		
G&A	$306,936																		
Total Operating Cost	$2,613,745																		
Royalty	$112,210																		
Property Taxes	$3,665																		
Salvage Value	-$26,524		-$3,979	-$3,183	-$3,183	-$2,918	-$2,652	-$2,652	-$2,122	-$2,122	-$1,857	-$1,857							
Reclamation/Closure																			
Production Cost	$2,703,096		-$3,979	-$3,183	-$3,183	-$2,918	-$2,652	-$2,652	-$2,122	-$2,122	-$1,857	-$1,857							
Net Operating Income	$4,295,624		$3,979	$3,183	$3,183	$2,918	$2,652	$2,652	$2,122	$2,122	$1,857	$1,857							
Depreciation																			
Initial Capital	$970,254																		
Sustaining Capital	$154,385	$6,533	$6,419	$7,111	$6,278	$5,324	$4,955	$4,749	$4,388	$3,743	$3,351	$3,434	$3,263	$2,614	$1,937	$1,616	$1,393	$975	$1,085
Total Depreciation	$1,124,639	$6,533	$6,419	$7,111	$6,278	$5,324	$4,955	$4,749	$4,388	$3,743	$3,351	$3,434	$3,263	$2,614	$1,937	$1,616	$1,393	$975	$1,085
Net Income after Depreciation	$3,170,985	-$6,533	-$2,440	-$3,928	-$3,096	-$2,406	-$2,303	-$2,097	-$2,266	-$1,622	-$1,494	-$1,577	-$3,263	-$2,614	-$1,937	-$1,616	-$1,393	-$975	-$1,085
Idaho Mine License Tax	$21,516																		
Idaho Corporate Income Tax	$149,737																		
Federal Income Tax	$655,807																		
Net Income after Taxes	$2,343,925	-$6,533	-$2,440	-$3,928	-$3,096	-$2,406	-$2,303	-$2,097	-$2,266	-$1,622	-$1,494	-$1,577	-$3,263	-$2,614	-$1,937	-$1,616	-$1,393	-$975	-$1,085
Cash Flow																			
Net Operating Income	$4,295,624		$3,979	$3,183	$3,183	$2,918	$2,652	$2,652	$2,122	$2,122	$1,857	$1,857							
Working Capital																			
Account Receivables		$19,005																	
Accounts Payable		-$7,102																	
Inventory (Parts)				$15,000															
Total Working Capital	$0	$11,904		$15,000															
Capital Expenditures																			
Initial Capital	$970,254																		
Sustaining Capital	$154,385	$8,013	$11,019	$7,234	$3,703	$3,112	$3,073	$3,035	$2,806	$2,771	$4,177	$4,111	$1,716	$332	$249	$1,190			
Total Capital Expenditures	$1,124,639	$8,013	$11,019	$7,234	$3,703	$3,112	$3,073	$3,035	$2,806	$2,771	$4,177	$4,111	$1,716	$332	$249	$1,190			
Cash Flow before Taxes	$3,170,985	$3,890	-$7,040	$10,949	-$520	-$194	-$421	-$382	-$685	-$649	-$2,320	-$2,255	-$1,716	-$332	-$249	-$1,190			
Cummulative Cash Flow before Taxes		$3,177,990	$3,170,949	$3,181,898	$3,181,378	$3,181,184	$3,180,763	$3,180,380	$3,179,696	$3,179,047	$3,176,727	$3,174,473	$3,172,757	$3,172,424	$3,172,175	$3,170,985	$3,170,985	$3,170,985	$3,170,985
Taxes	$827,060																		
Cash Flow after Taxes	$2,343,925	$3,890	-$7,040	$10,949	-$520	-$194	-$421	-$382	-$685	-$649	-$2,320	-$2,255	-$1,716	-$332	-$249	-$1,190			
Cummulative Cash Flow after Taxes		$2,350,929	$2,343,889	$2,354,838	$2,354,318	$2,354,124	$2,353,703	$2,353,320	$2,352,636	$2,351,987	$2,349,667	$2,347,412	$2,345,696	$2,345,364	$2,345,115	$2,343,925	$2,343,925	$2,343,925	$2,343,925

Economic Indicators before Taxes		
NPV @ 0%		$3,170,985
NPV @ 5%	5.0%	$1,954,725
NPV @ 7%	7.0%	$1,612,742
NPV @ 10%	10.0%	$1,205,447
IRR		31.9%
Payback	Years	2.2

Economic Indicators after Taxes		
NPV @ 0%		$2,343,925
NPV @ 5%	5.0%	$1,413,808
NPV @ 10%	10.0%	$836,479
IRR		27.0%
Payback	Years	2.5